

Management of Dry Flue Gas Desulfurization By-Products in Underground Mines

**Quarterly Report
April - June 1995**

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S. Esling
B. Paul
H. Sevim
E. Thomasson
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July 1995

Work Performed Under Contract No.: DE-FC21-93MC30252

For
U.S. Department of Energy
Office of Fossil Energy
Morgantown Energy Technology Center
Morgantown, West Virginia

MASTER

By
Southern Illinois University at Carbondale
Carbondale, Illinois

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BY-PRODUCTS IN UNDERGROUND MINES

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Department of Mining Engineering
Carbondale, Illinois 62901

July 1995

MANAGEMENT OF DRY FLUE GAS DESULFURIZATION BY-PRODUCTS IN UNDERGROUND MINES

Technical Progress Report -- April 1- June 30, 1995

CONTENTS

Section I	
Introduction and Summary.....	1
Section II	
Environmental Characterization.....	4
Dr. Bradley Paul - Co-Principal Investigator	
Section III	
Materials Handling & System Economics.....	11
Dr. Hasan Sevim - Co-Principal Investigator	
Section IV	
Environmental Assessment & Geotechnical Stability.....	14
Dr. Steven Esling - Co-Principal Investigator	
Section V	
Geotechnical Studies.....	16
Dr. Deepak Dutta - Co-Principal Investigator	
Section VI	
Hydraulic Placement.....	38
Dr. Xiao Yuan - Co-Principal Investigator	
Section VI	
Next Quarter.....	48
Appendix I	
Paper to be Presented at the Pittsburgh Coal Conference.....	50
Appendix II	
Packer Tests.....	57
Appendix III	
Paper to be Presented at the 8th International Symposium on Freight Pipelines	72
Appendix IV	
Geotechnical Core Log.....	79

**MANAGEMENT OF DRY FLUE GAS DESULFURIZATION
BY-PRODUCTS IN UNDERGROUND MINES**

Technical Progress Report -- April 1, - June 30, 1995

TABLES AND FIGURES

SECTION V

Table 1.	
Material Properties Used for Finite Element Modeling.....	23
Figure 1.	
Different Borehole over Pneumatic and Hydraulic Injection Panels...	18
Figure 2.	
Finite Element Mesh Used in the Model.....	20
Figure 3.	
Geologic Log Used for Finite Element Modeling.....	22
Figure 4.	
Maximum Floor Heave at the Entry Center.....	24
Figure 5.	
Maximum Roof Sag at the Entry Center.....	25
Figure 6.	
Percentage Reduction in Floor Heave Due to Backfilling.....	27
Figure 7.	
Percentage Reduction in Roog Sag Due to Backfilling.....	28
Figure 8.	
Amount of Pillar Punching Into the Floor.....	29
Figure 9.	
Maximum Compression Stress on the Backfill Material.....	30
Figure 10.	
Temperature Vs. Elapsed Time When Mixing in A Bowl Mixer.....	32
Figure 11.	
Temperature Vs. Elapsed Time When Simulating A Drum Type Mixer....	33
Figure 12.	
Typical Stress-Strain Curve for 7 Day Test.....	34

MANAGEMENT OF DRY FLUE GAS DESULFURIZATION BY-PRODUCTS IN UNDERGROUND MINES

Technical Progress Report -- April 1, - June 30, 1995

TABLES AND FIGURES

Section V (Continued)

Figure 13.	
Typical Streee-Strain Curve for 90-Day Test.....	35

SECTION VI

Figure 1.	
Cones for Slump Test.....	38
Figure 2.	
Curve of Slump Heights Vs Moisture Content.....	39
Figure 3.	
Relationship Between the Slump Heights by the Larger Cone and Those by the Small Cone.....	40
Figure 4.	
Relationship of Used Power with Moisture Content.....	41
Figure 5.	
Relation Between Stresses and Strains of Sample Group 1.....	42
Figure 6.	
Relationship Between Stresses and Strains of Sample Group 2.....	43
Figure 7.	
Relationship of Strength and Curing Time.....	44
Figure 8.	
Relationship of Young's Modulus and Curing Time.....	44
Figure 9.	
Relation of Stresses and Strains.....	45
Figure 10.	
Relation of Stresses and Strains.....	46

SECTION I

INTRODUCTION AND SUMMARY

MANAGEMENT OF DRY FLUE GAS DESULFURIZATION BY-PRODUCTS IN UNDERGROUND MINES

Technical Progress Report -- April 1 - June 30, 1995

INTRODUCTION AND SUMMARY

On September 30, 1993, the U.S. Department of Energy-Morgantown Energy Technology Center and Southern Illinois University at Carbondale (SIUC) entered into a cooperative research agreement entitled "Management of Dry Flue Gas Desulfurization By-Products in Underground Mines" (DE-FC21-93MC30252). Under the agreement Southern Illinois University at Carbondale will develop and demonstrate several technologies for the placement of coal combustion residues in abandoned coal mines, and will assess the environmental impact of such underground residues placement.

Previous quarterly Technical Progress Reports have set forth the specific objectives of the program, and a discussion of these is not repeated here. Rather, this report discusses the technical progress made during the period April 1 - June 30, 1995.

A final topical report on the SEEC, Inc. demonstration of its technology for the transporting of coal combustion residues was completed during the quarter, although final printing of the report was accomplished early in July, 1995. The SEEC technology involves the use of Collapsible Intermodal Containers (CIC's) developed by SEEC, and the transportation of such containers - filled with fly ash or other coal combustion residues - on rail coal cars or other transportation means. Copies of the final topical report, entitled "The Development and Testing of Collapsible Intermodal Containers for the Handling and Transport of Coal Combustion Residues" were furnished to the Morgantown Energy Technology Center.

The Rapid Aging Test columns were placed in operation during the quarter. This test is to determine the long-term reaction of both the pneumatic and hydraulic mixtures to brine as a leaching material, and simulates the conditions that will be encountered in the actual underground placement of the coal combustion residues mixtures. The tests will continue for about one year.

The injection wells at the Peabody Mine Number 10 site were drilled and completed during the quarter, as was a core well. The core well was continuously cored to a depth of 355.5 feet, and the core delivered to the Illinois State Geological Survey for study. The injection wells were studied with a borehole camera. After the core was removed, the core well was plugged and abandoned.

MANAGEMENT OF DRY FLUE GAS DESULFURIZATION BY-PRODUCTS IN UNDERGROUND MINES

Technical Progress Report -- April 1 - June 30, 1995

INTRODUCTION AND SUMMARY (Continued)

Also, some sixty subsidence monuments were installed around the test site at the Peabody Number 10 mine. Unfortunately, about one-third of the monuments were destroyed by heavy traffic at the mine, and will have to be replaced. Also, the surface casing of one of the injection wells was damaged by traffic; however, it is believed that the damage is easily reparable.

A technical paper, "Economics of Coal Combustion Residue Transportation" was prepared by Dr. H. Sevim and S. Gwamaka. The paper will be presented at the Pittsburgh Coal Conference in September.

Planning continued for the meeting/workshop to be held at SIUC on September 26 and 27, 1995. the meeting will bring together the three universities - University of Kentucky, Southern Illinois University at Carbondale, and West Virginia University -- along with U.S. Department of Energy personnel. Each of the three universities hold complimentary cooperative research agreements with the Department of Energy, and the workshop/meeting will review in detail the ongoing work under each agreement.

SECTION II

ENVIRONMENTAL CHARACTERIZATION

DR. BRADLEY PAUL
CO-PRINCIPAL INVESTIGATOR

MANAGEMENT OF DRY FLUE GAS DESULFURIZATION BY-PRODUCTS IN UNDERGROUND MINES

Technical Progress Report -- April 1 - June 30, 1995

ENVIRONMENTAL CHARACTERIZATION

The following tasks have been performed during the past quarter.

1- Work on the rapid age columns is progressing. In the previous report, completion of the rapid age columns was reported. In this quarter, a review was made of the general groundwater quality typical of the deep brines in the test area. These brine compositions are the basis on which the synthetic brine used in the columns is based. Several of the columns are now in operation. Data available at this time is insufficient for meaningful analysis.

2- A decision was made by the project teams as to the composition of both the pneumatic and hydraulic mixes to be placed underground. The decision included the addition of a lime manufacturing scrubber sludge to improve setting and strength properties of the hydraulic mix. This material had not previously been tested. Peabody coal company is moving forward with obtaining the permits for placement of the two mixtures. As part of the effort, analysis on all mix components and the proposed mix compositions have been made for all parameters specified by the Illinois Environmental Protection Agency. The results of these TCLPs are included in the data analysis section of this report.

3- The ASTM column leaching work has been started and results for some of the mix components are available and included in this report. The ASTM column leaching assembly was modified to permit the operation of up to six columns at a time instead of two. This should accelerate the pace for completion of all ASTM column work.

4- An inexpensive data recording computer has been purchased and placed in service by the Carterville laboratories.

At present it appears likely that all work for the characterization area will be available and complete by the end of the first phase of the project. Delays in obtaining the final mix proportions have resulted in a slow start for the rapid age testing. Two options are available for dealing with the rapid age delay. The results could be reported on schedule at the end of phase I with a shorter base-line time than in the original proposal. The second and preferred option is to give preliminary results at the end of phase I and then continue operation of the aging columns during the next six months of phase II while the surface demonstrations were in progress. Allowing the rapid age assembly to operate during phase II will not impact the surface demonstration. The rapid age work will furnish long term durability and leaching data that cannot be obtained from the other shorter term characterization and permitting tests.

MANAGEMENT OF DRY FLUE GAS DESULFURIZATION BY-PRODUCTS IN UNDERGROUND MINES

Technical Progress Report -- April 1 - June 30, 1995

ENVIRONMENTAL CHARACTERIZATION (CONTINUED)

Results and Discussion

A new material has been added to the hydraulic mixture. This material constitutes 5% of a mix containing 40% fly ash and 55% force oxidized scrubber sludge. The 5% additional material is itself a SO₂ removal scrubber sludge and does not reduce the percentage of SO₂ control byproducts from that proposed in earlier reports. The new scrubber sludge comes from a Venturi type scrubber used on a lime kiln. Because only the highest purity limestone is used in lime manufacture, the inerts content in the industrial scrubber sludge is low. The principle components of the material are hydrated calcium oxide, calcium carbonate, calcium sulfate and calcium sulfite. The kiln is fired by high sulfur Illinois basin coal.

TCLP test results are presented in Tables I and II. The final column reports the theoretical detection limit for the analytical technique being used. Elements not detected are reported as less than the detection limit rather than zero. All concentration values are reported in parts per million. Part per million units do not apply to pH, calcium carbonate equivalent (reported as a percentage of neutralizing power of pure calcium carbonate), or conductivity (reported in micro-mhos).

Permitting decisions for this material probably will be made relative to groundwater standards. The closer water bearing units at the Peabody #10 site are located about 12 feet above the coal. These units are of low permeability and contain brines. Surface groundwater resources apparently are not in communication with the deep brines since there is no detectable contamination of the near surface water resources. No data on the trace element content of the deep brines is available, but the salt content of the brines is generally around 2 to 4%. The water resources in the most apparent risk are not of class I quality, so comparisons in this analysis will be based on State of Illinois Class II groundwater standards.

There are few issues involved with the quality of the TCLP leachate relative to class II groundwater standards. These are discussed below.

1- The ADM FBC fly ash and spent bed and the CWLP fly ash and the lime waste all produce end point pH values in excess of the 9 limit often used for mine sites. It is known from long term column testing done for other research projects that the high pH values do not persist and come below the 9 limit. It is also interesting to note that none of the tests exceeded the 12.5 RCRA limit. The Dallman scrubber sludge had an endpoint pH of 5.29 which is below the mine discharge limit of 6. It should be remembered that the TCLP is buffered at pH 4.9 and the calcium carbonate equivalent value and the alkalinity value leave no doubt that this is an alkaline material. The endpoint pH for the hydraulic mix is inside the 6 to 9 limit. The mixture of spent bed and FBC fly ash is of course not within the 6 to 9 limit.

MANAGEMENT OF DRY FLUE GAS DESULFURIZATION BY-PRODUCTS IN UNDERGROUND MINES

Technical Progress Report -- April 1 - June 30, 1995

ENVIRONMENTAL CHARACTERIZATION (CONTINUED)

2- The total dissolved solids exceed the class II groundwater limit for all mix components and all mixtures. Since the brines in the region generally run about 2 to 4% dissolved solids and all the TCLP tests came out with a fraction of a percent dissolved solids, it is unlikely that groundwater quality will be deteriorated on this parameter.

3- Only the industrial boiler scrubber sludge did not exceed the class II groundwater limit for sulfate in the TCLP. All other components and mixes exceeded the sulfate limit. Of course such a result is almost inherent in any SO₂ control product when the class II limit is only 400 ppm. The anions for the deep brines in the region have not all been determined, but with the dissolved solids already as high as 2 to 4%, mostly with salt like cations, it is likely that sulfate is already above class II standards and any exceedance of the standard will already be from natural causes. Sulfate values in the leachate are high enough that is conceivable that the exact composition of the deep brines may have to be tested. Since the monitoring wells are now being installed to provide baseline data, the sulfate values will probably be available by next report.

4- The class II standard for selenium is 50 ppb. Although selenium was not detected in this round of tests, the detection limit is 75 ppb. It is apparent that selenium should not be a significant problem.

5- Arsenic concentrations were above the class II groundwater limit for both Dallman fly ash and the industrial scrubber sludge. The mixtures proposed for actual placement were all within class II limits so no problem is expected.

6- The Dallman fly ash is significantly above the class II limit for boron, and the hydraulic mixture thus also violates the class II limit. The basis for boron limits is plant toxicity. Concern about poisoning sensitive plant species with salt water in a low grade aquifer 400 feet deep seem unwarranted, and if a variance request is necessary, it could be granted without threat to the environment. Of course sea water already contains high levels of boron and it has been found in other work done at SIUC that many limestones in the region are high in boron. It is quit likely that deep brines that have been in extended contact with boron rich rocks will already violate class II limits from natural causes.

In the setting where it is proposed to place the FGD byproduct mixes there seems little reason to believe that any of the proposed mixes could pose a threat to the environment. Although there are some parameters that could be raised as a concern in the permitting process it seems unlikely that any issue will arise that cannot be dealt with by simply establishing the existing background water quality.

MANAGEMENT OF DRY FLUE GAS DESULFURIZATION BY-PRODUCTS IN UNDERGROUND MINES

Technical Progress Report -- April 1 - June 30, 1995

ENVIRONMENTAL CHARACTERIZATION (CONTINUED)

ASTM column tests have been partially completed for a number of the individual materials to be used in the mixes. The ASTM columns should furnish insights not available from any of the shake tests for several reasons.

1- The ASTM column test uses a nitrogen atmosphere and simulates leaching in an oxygen depleted environment below the water table. According to measurements taken in the bore-holes drilled into the test panels at the Peabody #10 mine site, the oxygen in the test panels is almost completely depleted. The mine strata is already below several low grade aquifer layers, though the mine itself is dry.

2- The ASTM column uses a more realistic solid to liquid ratio. With the mine itself dry and only enough water added to the materials to promote paste flow or set-off hydration reactions it is clear that shake tests using 20 parts water to every part solid are unrealistic. The ASTM column uses about 8 parts solid to 1 part liquid. Shake tests use a high ratio of water to solid to avoid common ion effects. With the water in the nearby strata at 3 to 4% solids it is clear that common ion effects and even ion exchange reactions will dominate groundwater reactions in the Peabody #10 mine.

3- The high ratio of solids to liquids in the ASTM column tend to promote higher concentrations of elements in the leachate. While most elements can be detected in 10s of parts per billion with ICP, concentrations must be over 10 times this high before good quantification is possible according the American Public Health Association. ASTM columns generally produce higher concentrations of trace elements in the leachate, improving the chances that a potentially leaching element will be detected and quantified within accuracy limits commonly expressed for ICP and AA techniques.

4- The ASTM column test produces leachate samples over time. One of the faults of shake tests is that they contact fresh material with solution, measure element concentrations, and assume that the concentrations seen are representative of the leaching of the bulk material. This is not the case with FGD byproducts and associated combustion residues. Many of the most noted toxic elements are volatilized at low temperature and are fumed out over the surface of particles exposed to the combustion exhaust gasses. Shake tests on fresh material leach unrepresentative surfaces and once the surface trace elements are leached away they will not be available in the bulk material in such concentrations again. Figure 1 represents the boron content of several consecutive ASTM column leach samples taken after 1, 2, 4, and 16 days. The pattern seen is highly representative of the leaching of volatile toxics from the surfaces of combustion residues.

MANAGEMENT OF DRY FLUE GAS DESULFURIZATION BY-PRODUCTS IN UNDERGROUND MINES

Technical Progress Report -- April 1 - June 30, 1995

ENVIRONMENTAL CHARACTERIZATION (CONTINUED)

At this point ASTM column tests are only complete for the FBC fly ash and spent bed used in the pneumatic mixes, and the force oxidized scrubber sludge used in the hydraulic mix. Most of the observations have more to do with what was not found than any problems found.

1- Most heavy trace metals were either not detected at all in the ASTM column leachates, or were just barely detected. With the concentration boost expected in ASTM columns this absence of trace heavy metals is a good sign that the materials pose no danger to the groundwater. Elements not found in concentrations high enough for good quantification or not found at all include

a- FBC Fly Ash, Ag, Be, Cd, Co, Cr, Cu, Hg, Mg, Ni, Pb, V.

b- FBC Spent Bed, Ag, Be, Co, Cr, Cu, Hg, Mg, Ni, Pb, Sb, V, Zn.

c- Force Oxidized FGD, Ag, As, Be, Cd, Co, Cr, Cu, Ni, Pb.

2- Amongst elements that were detected, most were present in concentrations that even at their peak would not produce groundwater violations even if no dilution or attenuation occurred. These elements include

a- FBC Fly Ash, Al, B, Ba, Fe, Mn, Mo, Si, V, Zn.

b- FBC Spent Bed, Al, Ba, Fe, K, Mn, Mo, Na, Si.

c- Force Oxidized FGD, Al, Fe, K, Mg, Mn, Mo, Si, V, Zn.

3- The only elements that leached in high concentrations were those typical of dissolved solids such as Ca, K, Na. Conductivity measurements indicate dissolved solids declined with time and never approached levels typical of the brines in the area.

4- Of the elements that were detected in concentrations high enough to produce class II groundwater violations, most occurred only as spike values or declined within less than 16 days to comply with class II standards. This means that leaching of these elements is unlikely to be persistent enough to cause the water in the nearest water bearing layers to violate class II standards. These elements included

a- FBC Fly Ash, As, Sb

b- FBC Spent Bed, As, B, Cd

MANAGEMENT OF DRY FLUE GAS DESULFURIZATION BY-PRODUCTS IN UNDERGROUND MINES

Technical Progress Report -- April 1 - June 30, 1995

ENVIRONMENTAL CHARACTERIZATION (CONTINUED)

c- Force Oxidized FGD, B, Sb, Se

The only element that seemed to leach with some persistence at concentrations in excess of class II standards was selenium from the FBC material. The concentrations found for selenium were below the levels needed for good ICP quantification, and the hydride system together with the EDL system, are being readied for more precise measurements. For such deep water units it should not be difficult to obtain a variance for selenium, even if it proves a problem. The ICP data is good enough to be certain that dilution or attenuation of less than 1 order of magnitude should bring water bearing units into the area into compliance unless selenium is already in violation from natural causes.

Problems and Remedial Action Proposed

The environmental characterization area has only two problems at this time. The first is a late start with the rapid aging tests. The solution to this problem was proposed earlier in this report. The second problem is that precise sulfate measurements are badly needed. Sulfate measurements are called for on all permit applications and it appears from the TCLP data that sulfate levels may become an issue in all test data. A liquid ion chromatograph is available, but functions only with a strip chart recorder. A computer is available in the water quality laboratory for quality control on ICP data, and this computer could be used to collect, integrate, and interpret the analog signal from the ion chromatograph. However, in order for this to be accomplished, and interface (between the chromatograph and computer) is necessary. Consideration is being given to acquire the interface.

SECTION III

MATERIALS HANDLING
AND
SYSTEM ECONOMICS

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MANAGEMENT OF DRY FLUE GAS DESULFURIZATION BY-PRODUCTS IN UNDERGROUND MINES

Technical Progress Report -- April 1 - June 30, 1995

MATERIAL HANDLING AND SYSTEM ECONOMICS

The following activities were undertaken in this quarter:

1. **Meeting with SEECTM Inc. staff:** A meeting was held between Dr. Sevim and the SEEC Inc.'s staff in SEEC Inc.'s office in Minneapolis. A modified version of system configuration and operating scenario of collapsible intermodal containers (CICTM) was presented to Dr. Sevim by Cathy Sherin. Dr. Sevim in turn presented the economic evaluation software developed for the project and how it was applied to each transportation alternative. The environmental concerns and the intangibles associated with each alternative were discussed. Overall, it was a very productive meeting for both parties.
2. **Meeting with Illinois Central (IC) staff:** Two meetings were held with IC staff; one in Carbondale and another in Chicago. The main topic of discussion was the rail rates for different transportation modes and scenarios. The IC staff was extremely cooperative, and expressed willingness to provide rail rates for a large number of cases. This information will be very valuable for the improvement of the software. In Chicago/Harvey, the IC's intermodal and bulk transfer terminal were visited. At the intermodal terminal, the operation of overhead crane and piggy-packer were observed. At the bulk transfer terminal, the transloading of powdered material from a pressure differential rail car into a pneumatic truck was observed. All these pieces of equipment were part of the transportation alternatives that have been presented in previous progress reports. Another topic of discussion was the new and innovative ways of transporting the dry as well as wet residues. The IC staff was very receptive to the idea of further investigation in this area.
3. **Technical paper:** A technical paper entitled "Economics of Coal Combustion Residue Transportation" was written for presentation at the 12th International Pittsburgh Coal Conference. This paper will also be published in the proceedings of the conference. The three transportation alternatives; pneumatic trucks, CICs and pressure differential rail cars (PD-cars), that have been the subject of our investigation were presented in this paper. The modifications in system operations of CICs and PD-cars were incorporated in the analysis. "A copy of this paper is included in this report as Appendix III."
4. **Communication with Walton/Stout, Inc.:** This company specializes in designing and manufacturing pneumatic conveying systems for bulk material filling and emptying stations. After a meeting with their staff in Lithonia, Georgia, description of the transportation alternatives were sent to them, and information on filling and emptying stations for these alternatives were requested. A report on the preliminary analysis of these systems was recently sent by Walton/Stout. Further cooperation will be sought in the next quarter to obtain more specific information.

MANAGEMENT OF DRY FLUE GAS DESULFURIZATION BY-PRODUCTS IN UNDERGROUND MINES

Technical Progress Report -- April 1 - June 30, 1995

MATERIAL HANDLING AND SYSTEM ECONOMICS

5. Update of spreadsheets: The spreadsheets used in the economic evaluation were updated for a better presentation of the line items. A document describing the line items in these spreadsheets was also completed. In this document, the source of data entry, the assumptions behind the computations, and the step-by-step computations are described so that the reader can verify and understand the logic of the spreadsheets. A few more modifications will be made both in the appearance of the spreadsheets and in the description of the line items.

6. The interactive software: The window version of the economic evaluation program mentioned in the last quarterly report has been initiated. Mr. Da Lei, a graduate student working with Dr. Sevim, is making good progress in the development of the software. This version will be interactive and equipped with a number of menus to facilitate the evaluation of residue handling, transportation, and injection.

PLANS FOR THE NEXT QUARTER

The following activities are planned for the next quarter:

1. Finish updating the spreadsheets and the accompanying documents.
2. Consider the modifications in the system operations of CICs and PD-cars and conduct economic evaluations using the leasing option. Conduct sensitivity analysis on the rail haul rates to reveal the effect of this factor on the economics of the CICs and PD-cars.
3. Investigate and evaluate alternative transportation systems. Search for systems that can handle both the dry and wet residues. Explore the possibility of using grain cars in conjunction with the articulated chain conveyors of Wilson Manufacturing and Design Inc., mentioned in the last quarterly report. Explore the possibility of using "Coltainer" concept developed by Norfolk Southern Corp.
4. Make progress on the development of the interactive software.
5. Seek further cooperation from Walton/Stout Inc., to obtain engineering and cost data for residue filling and emptying stations.
6. Write a detailed description of the information that will be requested from the IC for rail haul rates and obtain that information as soon as possible.

SECTION IV

ENVIRONMENTAL ASSESSMENT
AND
GEOTECHNICAL STABILITY

DR. S. ESLING
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MANAGEMENT OF DRY FLUE GAS DESULFURIZATION BY-PRODUCTS IN UNDERGROUND MINES

Technical Progress Report -- April 1 - June 30, 1995

ENVIRONMENTAL ASSESSMENT AND GEOTECHNICAL STABILITY AND SUBSIDENCE IMPACTS

Accomplishments

Researchers from SIUC and the ISGS monitored coring at the Peabody 10 Mine June 21 through June 23. One continuous core was collected to a depth of 355.55 feet and returned to the ISGS for detailed description. The "Appendix II of this report contains a preliminary geotechnical log of the core prepared by Nelson Kawamura of the ISGS." Geophysical logs were taken of the borehole before it was sealed. SIUC personnel worked on developing a hydraulic injection system for the packer tests based on an ISGS design and familiarizing themselves with packer operation. The appendix contains a draft summary of packer test methodology prepared by Miquel Restrepo and Edward Mehnert of the ISGS. The critical review of packer testing should be considered a working draft that is subject to review and change.

The core log supports earlier findings that disposal at this site poses little risk to the environment. Shale dominates the section above the mine. Rare sandstone units appear to have low hydraulic conductivity. Screens for the monitoring wells will be set in the more permeable units. The number of monitoring wells was reduced from six, as proposed, to five. The hydraulic and pneumatic injection sites are located near each other, and can share one of the monitoring wells. Each hole drilled will contain a nest of wells (up to three) screened at different horizons in order to assess vertical hydraulic gradients.

Plans

Packer tests of all monitoring well holes, casing of the monitoring wells, and installation of data acquisition equipment will be completed next quarter. In addition, background groundwater quality samples will be collected from the five monitoring wells.

SECTION V

MIX CHARACTERIZATION,
NUMERICAL MODELING,
AND FIELD INSTRUMENTATION

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MANAGEMENT OF DRY FLUE GAS DESULFURIZATION BY-PRODUCTS IN UNDERGROUND MINES

Technical Progress Report -- April 1 - June 30, 1995

MIX CHARACTERIZATION, NUMERICAL MODELING, AND FIELD INSTRUMENTATION

SPECIFIC OBJECTIVES

The geotechnical characterization of mixes for pneumatic and hydraulic placements is intended for the determination of their short and long term strengths, elastic moduli, stress-strain curves, swelling and slump characteristics, linear expansions, heat of reactions, mass loss at extreme temperatures, and density. Hydraulic placement of CCR (coal combustion residues) and FGD by-products into abandoned mine workings may negatively impact the stability of mine workings and surface subsidence since Illinois coal seams are generally associated with thick (2-4 ft) and weak (300-1000 psi) floor strata. Short-term subsidence due to wet backfilling can cause damage to surface structures and impact land use patterns. In addition, the stability of underground bulkhead may be negatively impacted due to active pressures imposed by hydraulic or pneumatic placement of by-products.

The immediate floor strata associated with No. 6 coal seam (Herrin Seam) at Peabody No 10 demonstration mine are known to be weak (Chugh et al., 1989). Hence, objectives of the geotechnical assessment also include:

1. analysis of the stability of abandoned mine workings prior to and after disposal of combustion by-products;
2. estimation of the surface movements and their characteristics due to wet disposal of by-products;
3. assessment of the stability of isolation structures such as bulkhead to withstand pressures due to the disposal of FGD by-products;
4. monitoring of long term surface and sub-surface movements prior to and after backfilling of coal combustion by-products.

The results of the demonstration studies at Peabody No. 10 mine will be generalized for other areas within the Illinois Coal Basin.

MANAGEMENT OF DRY FLUE GAS DESULFURIZATION BY-PRODUCTS IN UNDERGROUND MINES

Technical Progress Report -- April 1 - June 30, 1995

MIX CHARACTERIZATION, NUMERICAL MODELING, AND FIELD INSTRUMENTATION

TASKS WORKED ON DURING THE QUARTER

During this quarter the drilling of two injection wells (one for hydraulic injection and the other for pneumatic injection) at Peabody #10 mine was completed. A borehole camera was used to observe the condition of the wells and underground openings. Sixty surface subsidence monuments were installed to measure vertical surface movements. Finite element modeling was carried out to study the effects of mix placements on roof sag, floor heave, and pillar punching into the floor. Additionally the characterization of pneumatic and hydraulic mixes was continued.

Injection Wells and Underground Conditions

Figure 1 shows different boreholes proposed to be drilled over pneumatic and hydraulic injection panels (Figure 1 does not show any water monitoring well). Drilling of pneumatic and hydraulic injection wells has been completed.

Observations by borehole camera in the pneumatic injection hole indicate dry borehole walls. The hydraulic injection hole indicates seepage of water from the sandstone unit 15 to 30 ft above the roof-line. The heights of the openings in both panels are approximately 6.75 ft. The pillars in the pneumatic panel do not show any sign of spalling and the old timber posts appear to be in good shape. The immediate roof of the pneumatic panel is limestone. The pillars in the hydraulic panel show signs of spalling and the old timber posts are buckled, indicating they are under load. The immediate roof of the hydraulic panel is 2.5 ft thick shale. Openings in both panels are in good conditions.

Oxygen in the air coming through the boreholes from the panels was 3%, indicating high content of methane. Methane in the underground air cannot be measured because of the low oxygen content in the air. No carbon monoxide in the underground air is detected. A complete analysis of underground air quality is planned for the future.

Installation of Subsidence Monuments

Sixty subsidence monuments have been installed over the pneumatic and hydraulic injections panels. However, due to the demolition works being carried out at Peabody #10 mine site, 20 monuments have been destroyed. Replacement monuments will be installed in the next quarter. It is intended to take the baseline measurements after the demolition work at the injection sites is completed.

MANAGEMENT OF DRY FLUE GAS DESULFURIZATION BY-PRODUCTS IN UNDERGROUND MINES

Technical Progress Report -- April 1 - June 30, 1995

MIX CHARACTERIZATION, NUMERICAL MODELING, AND FIELD INSTRUMENTATION

quarter. It is intended to take the baseline measurements after the demolition work at the injection sites is completed.

Finite Element Modeling

The causes of surface subsidence in the Illinois coal basin are the weakening of floor strata resulting in floor heaves and pillar punching into soft floors. This results in pillar and roof failures. The finite element modeling was conducted to determine the amounts of floor heave, roof sag, and pillar punching over a period of time with and without back filling. If the backfilling can reduce floor heaves, roof sags, and pillar punching, then it can reduce the subsidence on the surface. Also, reductions in floor heaves, roof sag, and pillar punching due to partial backfilling and backfilling with different material stiffnesses were investigated using finite element modeling.

Figure 2 shows the mesh of the two-dimensional, plane strain finite element model used for the study. The mesh is a vertical section with parallel entries and the crosscuts are parallel to the plane of the mesh. This implies that the entries are tunnel like openings and are infinitely long. The pillars are also treated as infinitely long in the perpendicular direction. This type of two-dimensional analyses ignore the effects of the crosscuts which result in pillar stresses that are less in analyses than in actuality. To circumvent this problem, Pariseau and Sorenson (1979) have suggested to increase the actual unit weight of the overburden strata by the factor $(1 + W_e/W_p)$ where W_e and W_p are the entry and pillar widths, respectively. This technique leads to pillar stresses from two-dimensional analyses that are within a few percent of three-dimensional results (Pariseau and Sorenson, 1979). In the current model, the three-dimensional effect is simulated by increasing the unit weight of overburden strata by the above mentioned factor.

The model is formulated as a large displacement, small strain problem and the geometric non-linearity is incorporated in the model. All the strata in the model are assumed to be linear elastic except the immediate floor. The immediate floor is assumed to have time dependent behaviors. An empirical creep equation of the form $e = as^bt^c$ (where e is the creep strain, s is the stress, t is the time in days, and a , b , c are constants) is used to simulate the time dependent behavior of the floor strata. Chugh et. al (1994) have done extensive testing of time dependent floor strata behaviors in the Illinois coal basin and developed a visco-elastic model for the time dependent floor behavior. Data from those test results were used to determine the creep constants a , b , and c . These constants are: $a = 2.44 \times 10^{-4}$, $b = 1.1$, and $c = 0.82$.

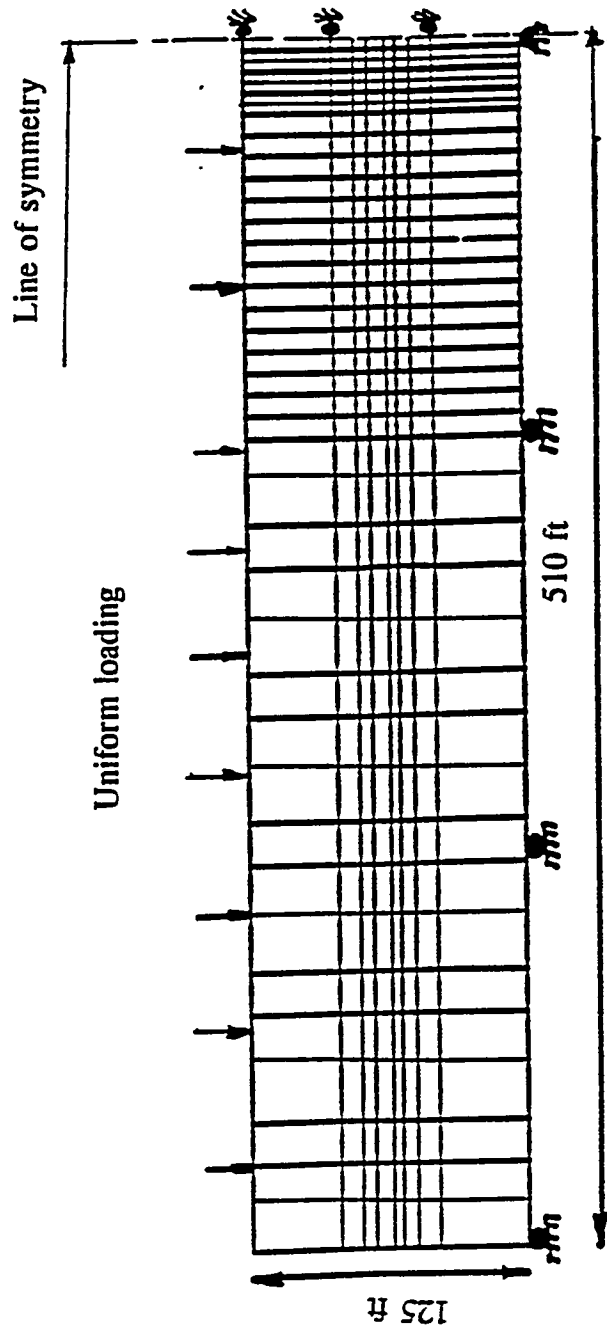


Figure 2. Finite Element Mesh Used in the Model

MANAGEMENT OF DRY FLUE GAS DESULFURIZATION BY-PRODUCTS IN UNDERGROUND MINES

Technical Progress Report -- April 1 - June 30, 1995

MIX CHARACTERIZATION, NUMERICAL MODELING, AND FIELD INSTRUMENTATION

The typical geologic log as shown in Figure 3 was used for modeling. The extraction height of the coal seam is seven ft and the immediate floor is 10 ft thick clay stone (weak floor). The immediate roof is four ft. thick gray shale overlain by an eight ft thick limestone stratum. In the model, 68 ft of overburden strata are considered and the rest of the overburden, up to the surface, is modeled by using vertical overburden loads. A floor thickness of 50 ft beneath the coal seam is used for modeling.

Though the model simulates two complete barrier pillars of 80 ft length and two half barrier pillars (40 ft length) at two extreme ends, only half of the mesh is used because of the symmetry (see Figure 2). A five entry panel configuration with 20 ft wide entries and 40 ft wide pillars is used in the model. The length of the actual mesh is 510 ft with one half barrier pillar (at the extreme left) and one full barrier pillar. The model is loaded on the top by uniform vertical loads to simulate 282 ft of overburden which corresponds to 350 ft of overburden thickness (282 ft of loading and 68 ft of actual overburden materials). The bottom of the model is constrained in the Y direction and allowed to move in the X direction. While the right hand side of the model is constrained in the X direction and allowed to move in the Y direction, a uniform displacement is prescribed in the left side of the model to simulate a horizontal stress of 1000 psi in limestones.

The region of interest is one half entry and one half pillar at the extreme right (see Figure 2) where the element size is 2 ft by 1 ft.

Table 1 shows material properties used for different strata and backfill materials for modeling purposes. The backfill material is assumed to have linear properties and four different values of Young's modulus are used (also shown as percentages of coal's Young's modulus in Table 1).

The modeling procedures involve two sets of steps--one without any backfill and one with a backfill material. The first set of steps involve:

1. Apply pre-mining stresses to the model without any opening.
2. Excavate all the openings simultaneously within 30 days to simulate formation of entries. Redistribution of stresses occur and vertical displacements in the roof and floor are determined due to the creation of openings.

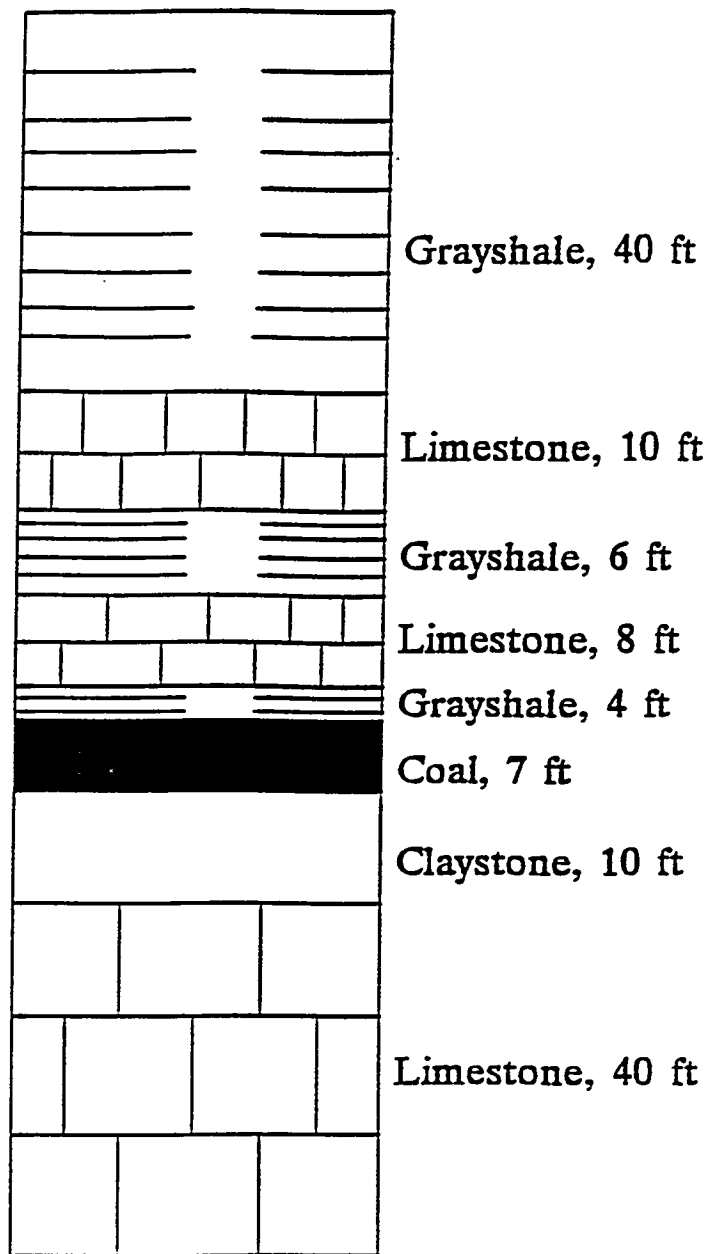


Figure 3. Geologic Log Used for Finite Element Modeling

MANAGEMENT OF DRY FLUE GAS DESULFURIZATION BY-PRODUCTS IN UNDERGROUND MINES

Technical Progress Report -- April 1 - June 30, 1995

MIX CHARACTERIZATION, NUMERICAL MODELING, AND FIELD INSTRUMENTATION

3. Run the model to simulate nine years at an increment of 30 days. Stresses and strains are updated at every increment. Time dependent strains induce time dependent floor, pillar and roof movements.

In the second set of steps, the excavations are created as before but the openings are backfilled after one year. The backfill material is born stress-free in the already deformed openings. The model is run for nine years and stresses and strains are updated at an increment of 30 days. Floor heaves, roof sags, and pillar punching are determined from vertical displacements.

Table 1 Material properties used for finite element modeling

Materials	Young's Modulus, psi	Poisson's Ratio	Unit Weight, pcf
Limestone	700,000	0.15	170
Gray shale	250,000	0.25	160
Coal	150,000	0.30	75
Claystone	25,000	0.35	160
Backfill materials	6,250 (4%)* 12,500 (8%)* 18,750 (12%)* 25,000 (16%)*	0.25	70

Note: * indicates percentage of coal's Young's modulus

Results and discussions Figures 4 and 5 show the maximum floor heave and roof sag at the entry centers for different E (Young's Modulus) values of the backfill material. Also shown is the case when partial backfilling is done (one foot of gap between the roof and the backfill material). The case of partial backfilling considered here is the worst case scenario where complete backfilling cannot be achieved at the injection borehole. In practice, however, complete backfilling is achieved in the region around the injection borehole and voids remain at the periphery of the panel.

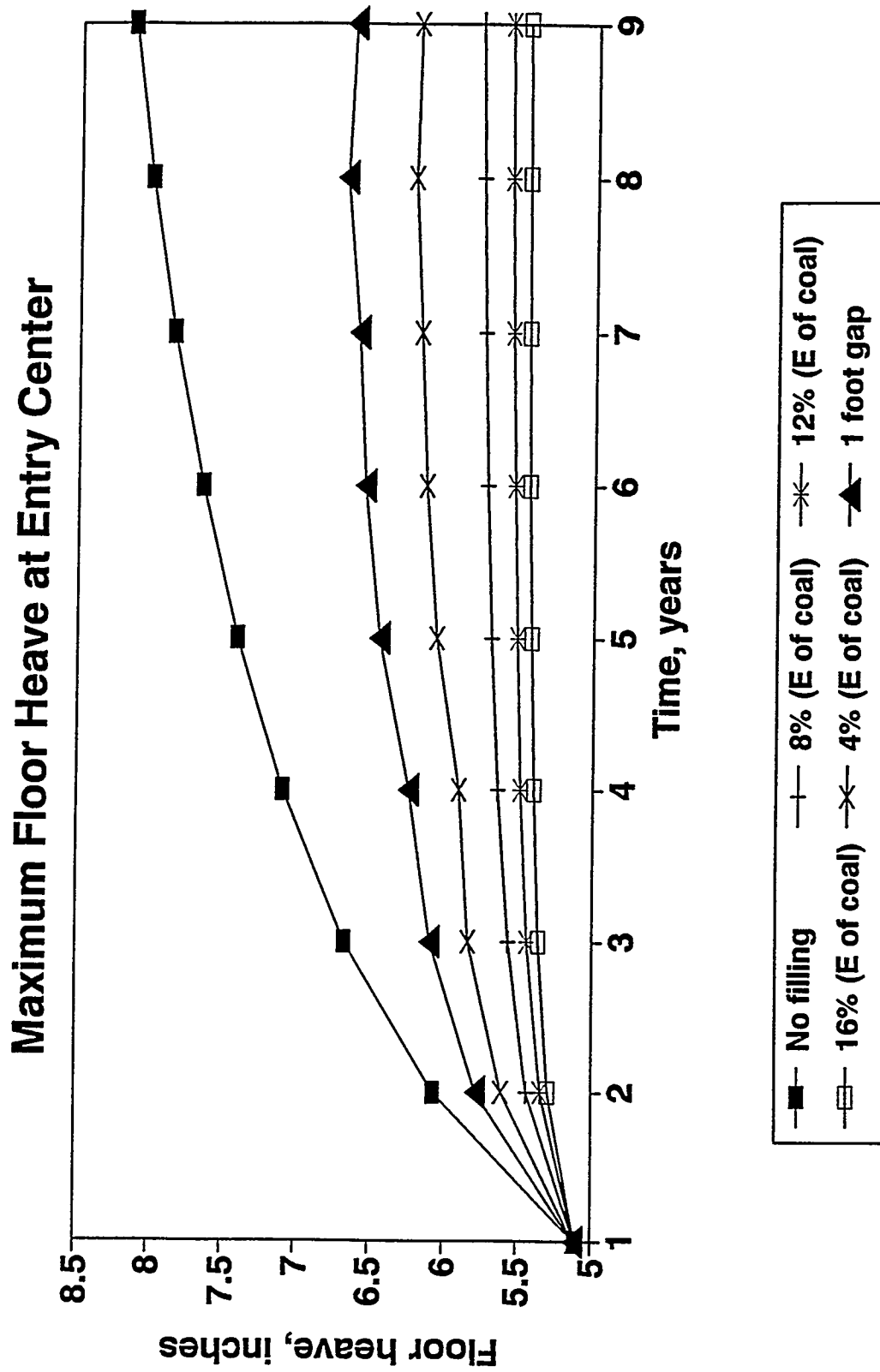


Figure 4 Maximum floor heave at the entry center

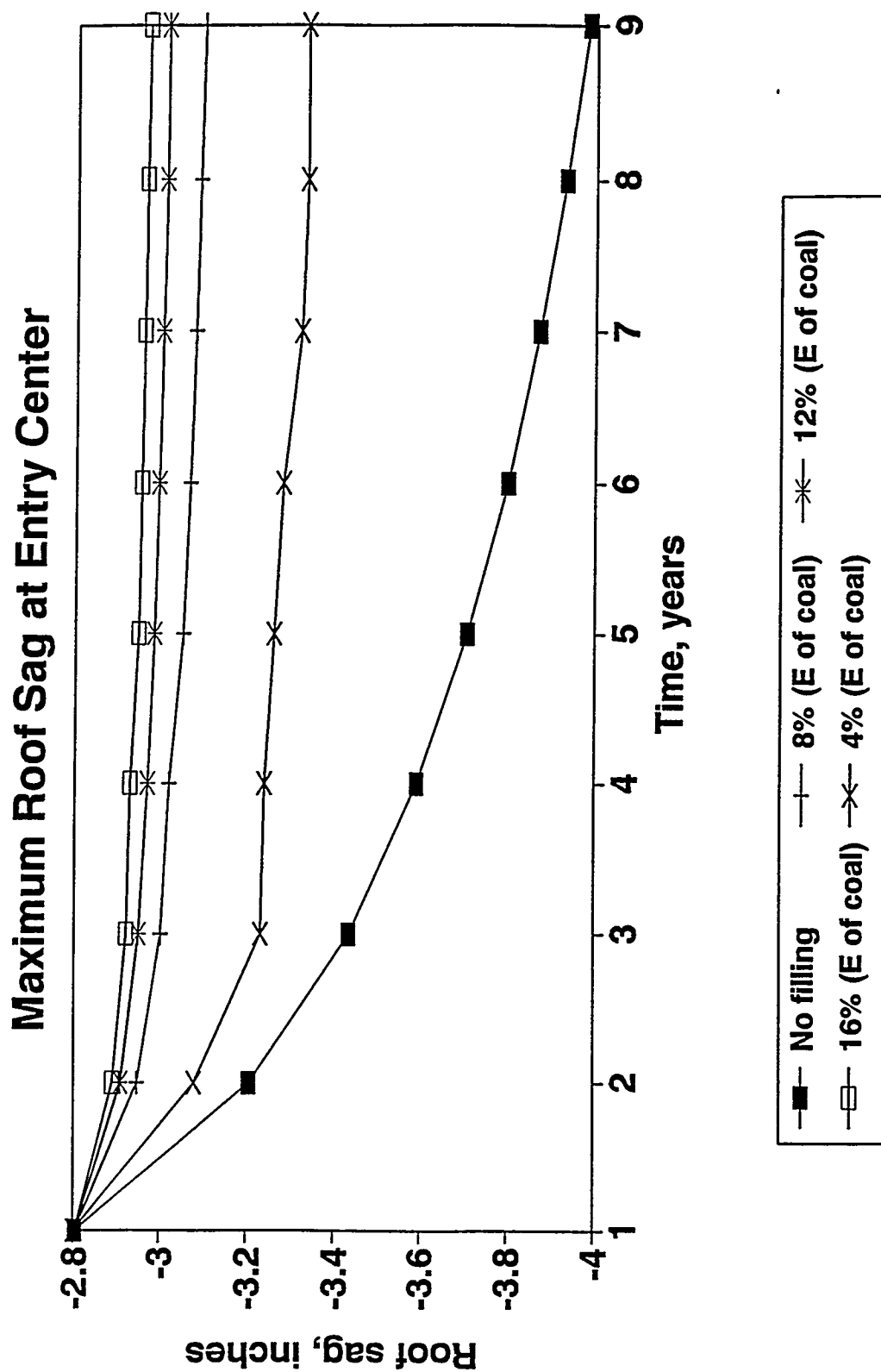


Figure 5 Maximum roof sag at the entry center

MANAGEMENT OF DRY FLUE GAS DESULFURIZATION BY-PRODUCTS IN UNDERGROUND MINES

Technical Progress Report -- April 1 - June 30, 1995

MIX CHARACTERIZATION, NUMERICAL MODELING, AND FIELD INSTRUMENTATION

Figures 6 and 7 show the percentage reduction in floor heaves and roof sag due to backfilling. The maximum floor heave after nine years without backfilling is 8.3 inches. If backfilling is done after one year when the maximum floor heave of 5.2 inches has already occurred, the floor heave increases by an amount 0.2 inches for a backfill material whose Young's modulus is 16% that of the coal. Figure 6 shows that this amounts to 88% reduction in floor heaves. For an extremely weak backfill material (whose Young's modulus is 4% that of the coal), the reduction in floor heave is 60% due to backfilling. In the worst case scenario when there is one foot gap between the backfill material and the roof, the reduction in floor heave is 48%. A similar reduction is obtained for roof sag as shown in Figure 7. Figure 8 shows the amount of pillar punching into the floor in the case of no backfilling, backfilling with different E values, and partial backfilling. The figure show that the amount of pillar punching can be significantly reduced by backfilling.

Figure 9 shows the maximum compressive stresses on the backfill materials for different E values of the back fill material. The figure shows that the maximum compressive stress on the backfill material is between 60 to 170 psi. It implies that not much stress redistribution occurs due to backfilling with weak backfill materials. The reduction of floor heaves, pillar punching, and roof sags can be attributed to the confinement provided by the backfill material. The failure of the backfill material when placed underground is negligible because the stresses on it is very low. The analyses show that a weak backfill material can effectively reduce floor heaves, roof sags, and the amount of pillar punching into the floor, which, in turn, can reduce surface subsidence.

Characterization of Pneumatic Mixes

The pneumatic mix is finalized to have either 80% FBC fly ash and 20% spent bed, or 70% FBC fly ash and 30% spent bed. Considering the current production of FBC fly ash and spent bed, a 80-20 (fly ash-spent bed) mix is more desirable. When water is added to the mix of FBC fly ash and spent bed, the heat of reaction raises the temperature to 99.5 degree centigrade. At this high temperature, the water in the mix vaporizes. It has been reported earlier that a nominal moisture content of 40% is reduced to an actual moisture content of approximately 18% by the time of sample preparation (see Annual Technical Progress Report for the Period October 1, 1993-September 30, 1994, Section III, Page 38). Similarly, a nominal moisture content of 30% is reduced to an actual moisture content of 10%-12%.

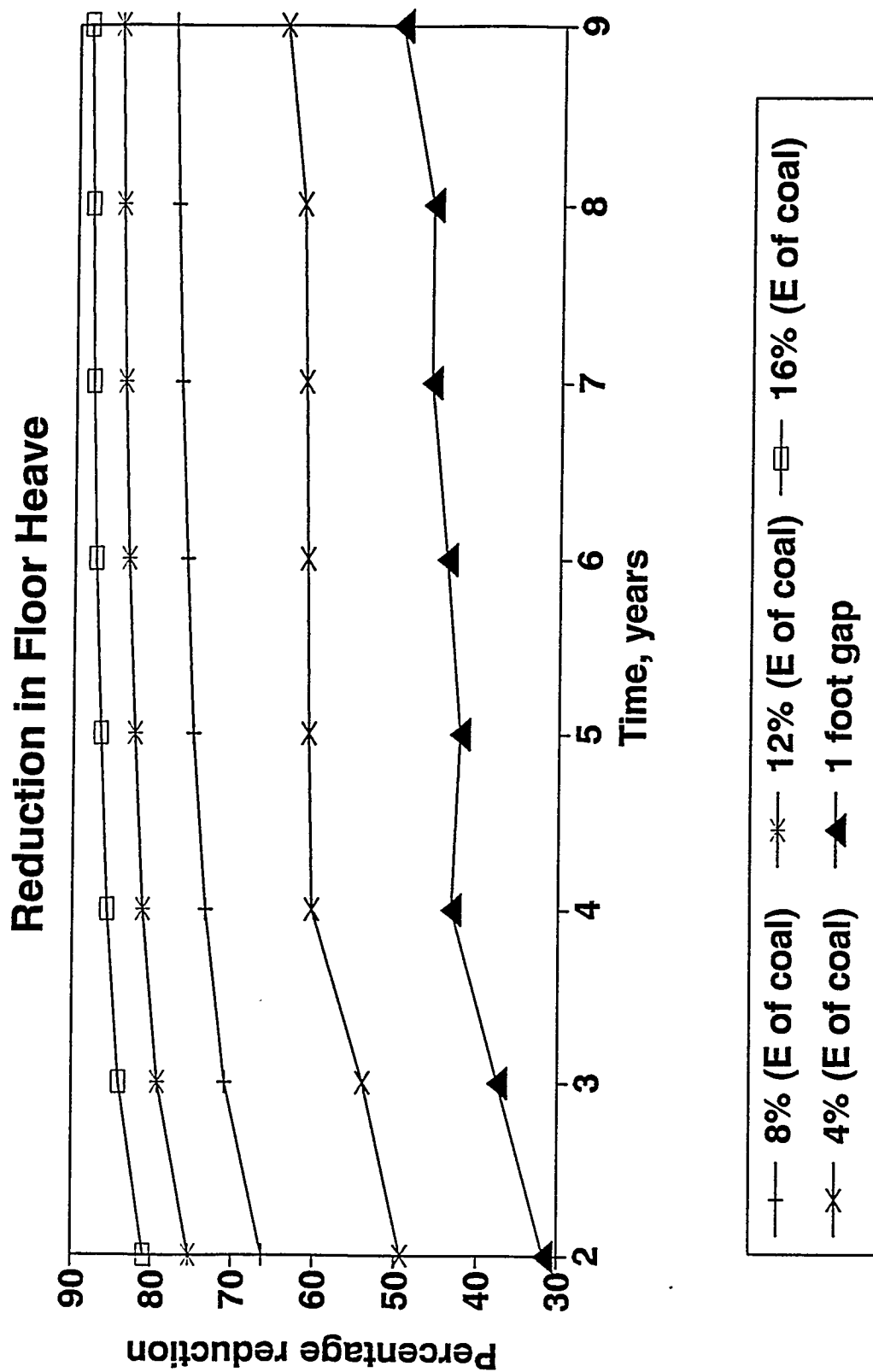


Figure 6 Percentage reduction in floor heave due to backfilling

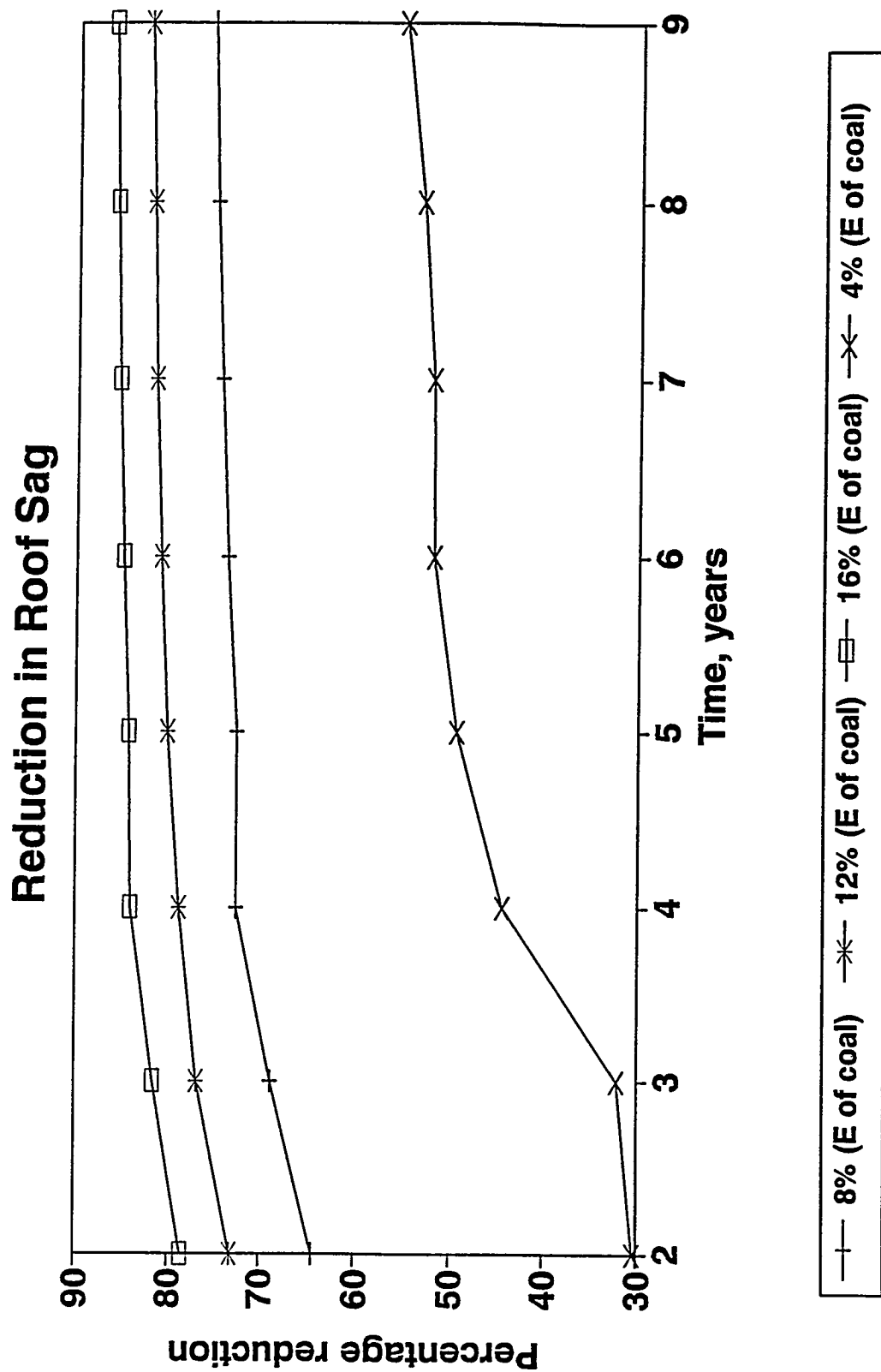


Figure 7 Percentage reduction in roof sag due to backfilling

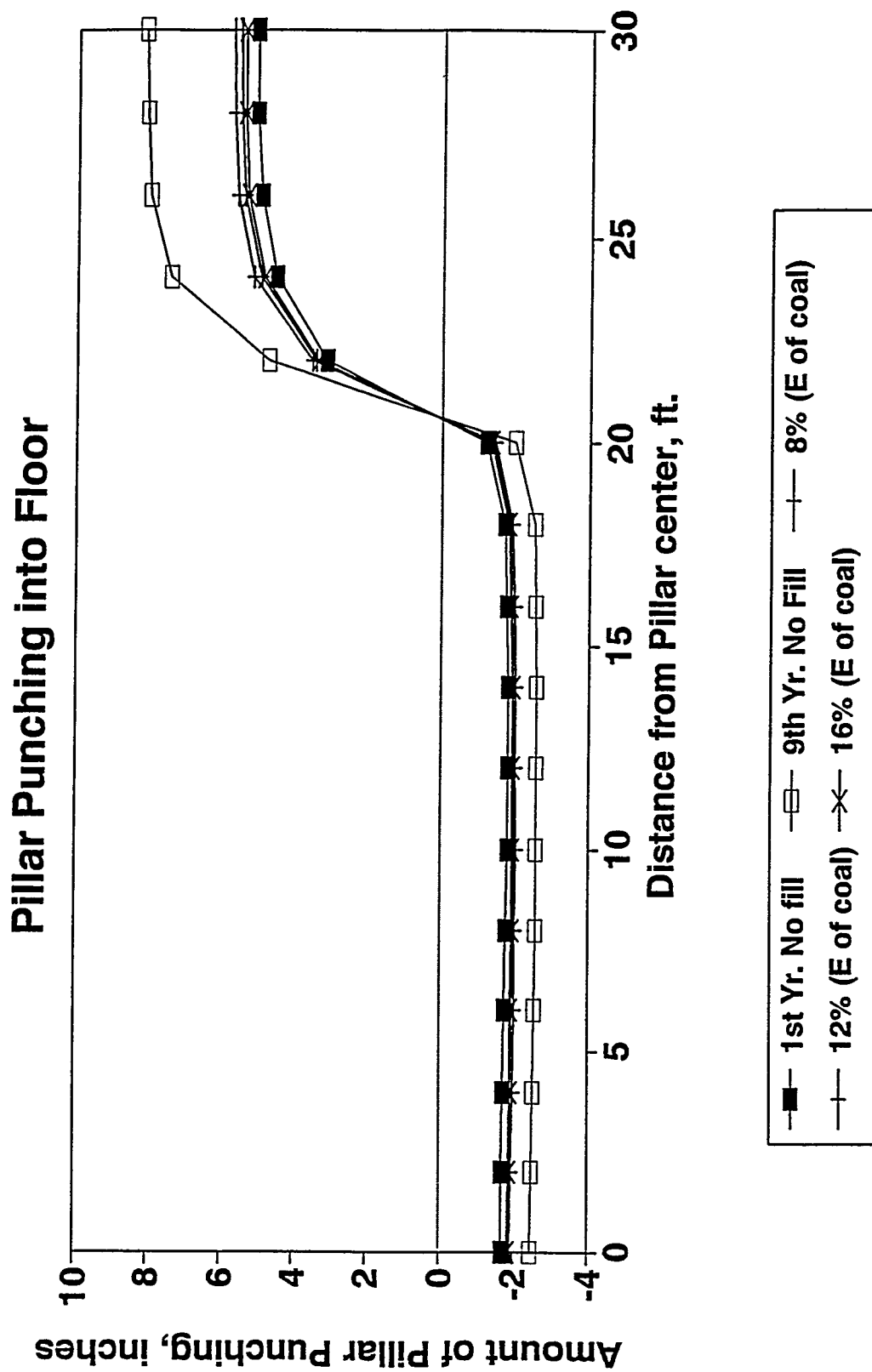


Figure 8 Amount of pillar punching into the floor

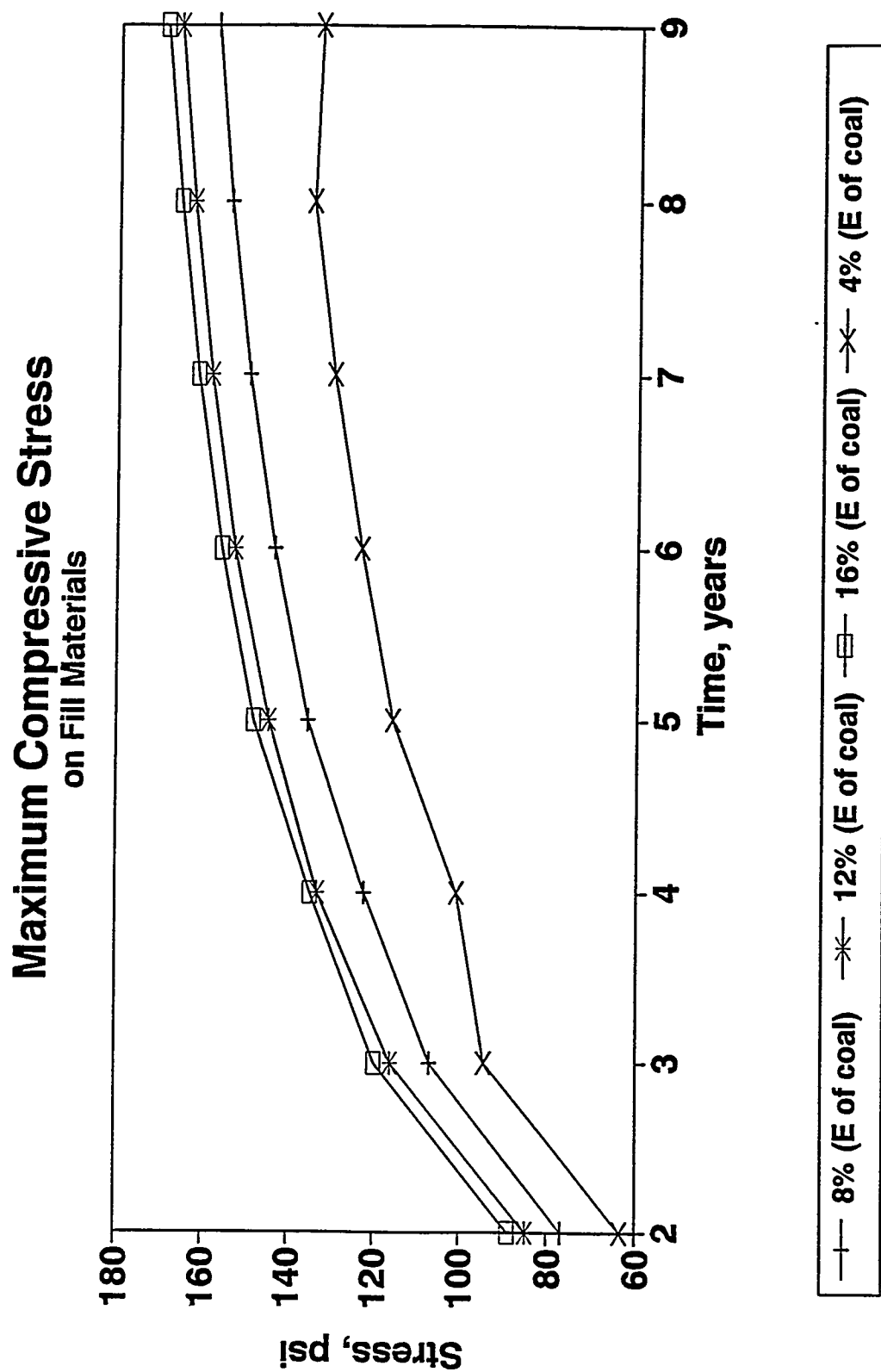


Figure 9 Maximum compressive stress on the backfill material

MANAGEMENT OF DRY FLUE GAS DESULFURIZATION BY-PRODUCTS IN UNDERGROUND MINES

Technical Progress Report -- April 1 - June 30, 1995

MIX CHARACTERIZATION, NUMERICAL MODELING, AND FIELD INSTRUMENTATION

Two tests were performed to study the drop of temperature and moisture with time. Figure 10 shows the drop of temperature and moisture with time when the mixing is done in a bowl mixer. A 30% nominal moisture content in the mix drops to 18% after five minutes of mixing (shown at time 0 in Figure 10) and eventually drops to 11% after 30 minutes. The temperature drops to 62 degrees centigrade from 99.5 degrees centigrade in the first 30 minutes. It takes two hours for the temperature to drop to the room temperature.

In the field, a drum type mixer (like a concrete mixer) will be used to mix FBC fly ash, spent bed, and water for pneumatic injection. The second test to study the drop of temperature and moisture was done by simulating a drum type mixer. In the drum type mixer, the material falls under its own weight and more surface area comes in contact with the air. This accelerates the temperature drop. Figure 11 shows the temperature and moisture drop when simulating a drum type mixer. As before, a 30% nominal moisture content drops to 18% after five minutes. At the same time, the temperature drops to 62 degrees centigrade. Because of the low temperature, the rate of moisture evaporation in the mix is reduced significantly. In 30 minutes, the temperature drops to the room temperature and the actual moisture content of the mix is 17%. A 10% moisture is added to the mix after 30 minutes. This does not increase the temperature further. The actual moisture content of the mix at 27% is blowable. This test suggests that 30% moisture should be added to the mix (FBC fly ash and spent bed) and mixed in a drum type mixer for five minutes. An extra 10% water should be added after five minutes. This will ensure an actual moisture content of 27% to 28% in the mix and the mix is blowable at this moisture content.

Uniaxial compressive strength and Young's modulus of pneumatic mixes Six three-inch diameter cylindrical samples with 80% FBC fly ash, 20% spent bed and 28% actual moisture content were tested to determine their 7-day and 90-day uniaxial compressive strengths and Young's moduli. Typical stress strain curves for seven-day and 90-day samples are shown in Figures 12 and 13, respectively. The seven-day compressive strengths of three samples were 28, 39, and 54 psi with an average of 40 psi. The average Young's modulus of the three samples was 3850 psi. The 90-day average compressive strength of the samples was 148 psi and the average Young's modulus was 8735 psi.

PLANS FOR THE NEXT QUARTER

Sample characterization for the pneumatic and hydraulic mixes will continue. Borehole instrumentation will start as soon as the drilling of boreholes are completed

Base line data on surface and subsurface movements will be taken and, thereafter, movement data will be collected at an interval of 30 days.

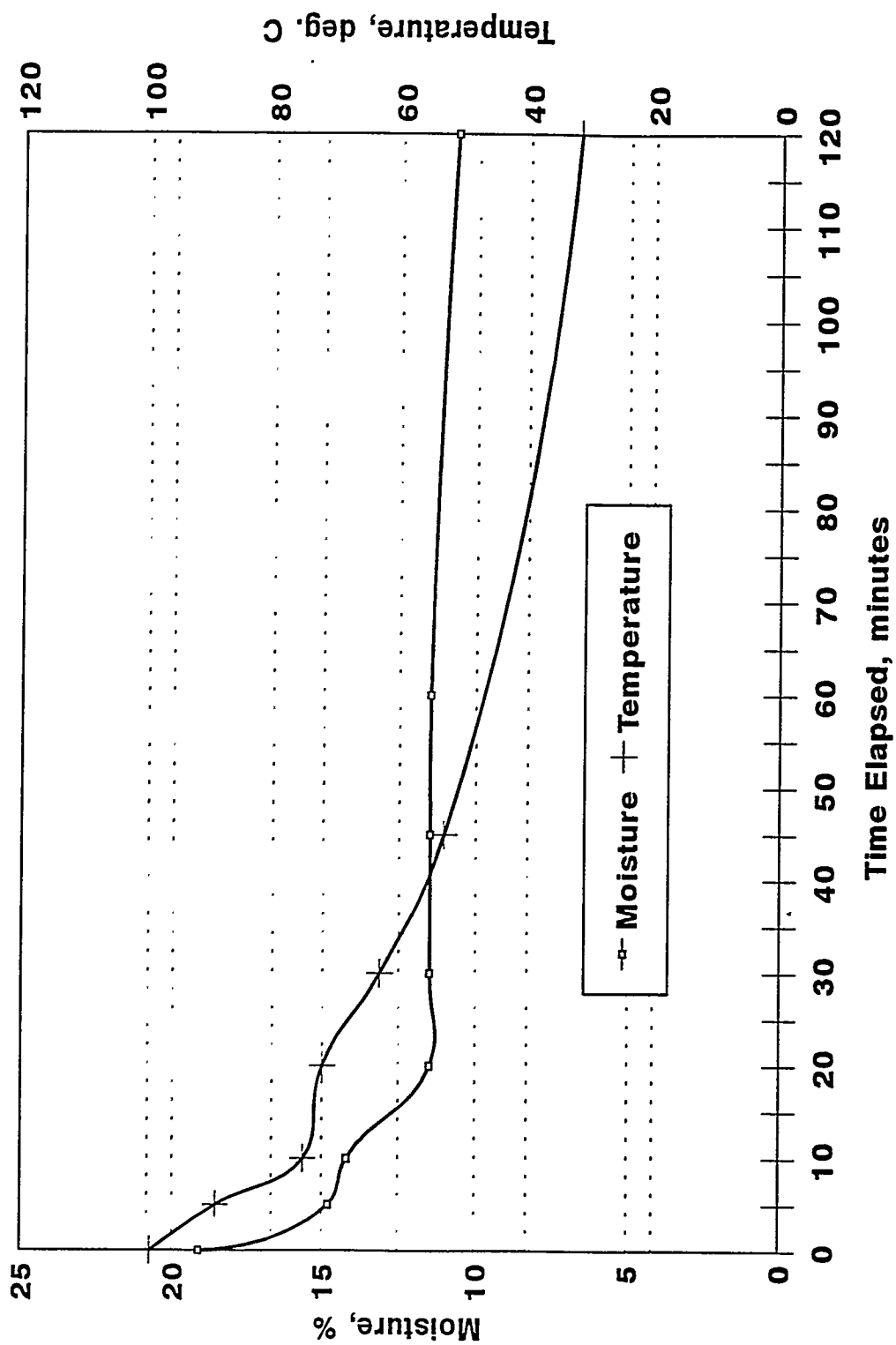


Figure 10 Temperature vs. elapsed time when mixing in a bowl mixer

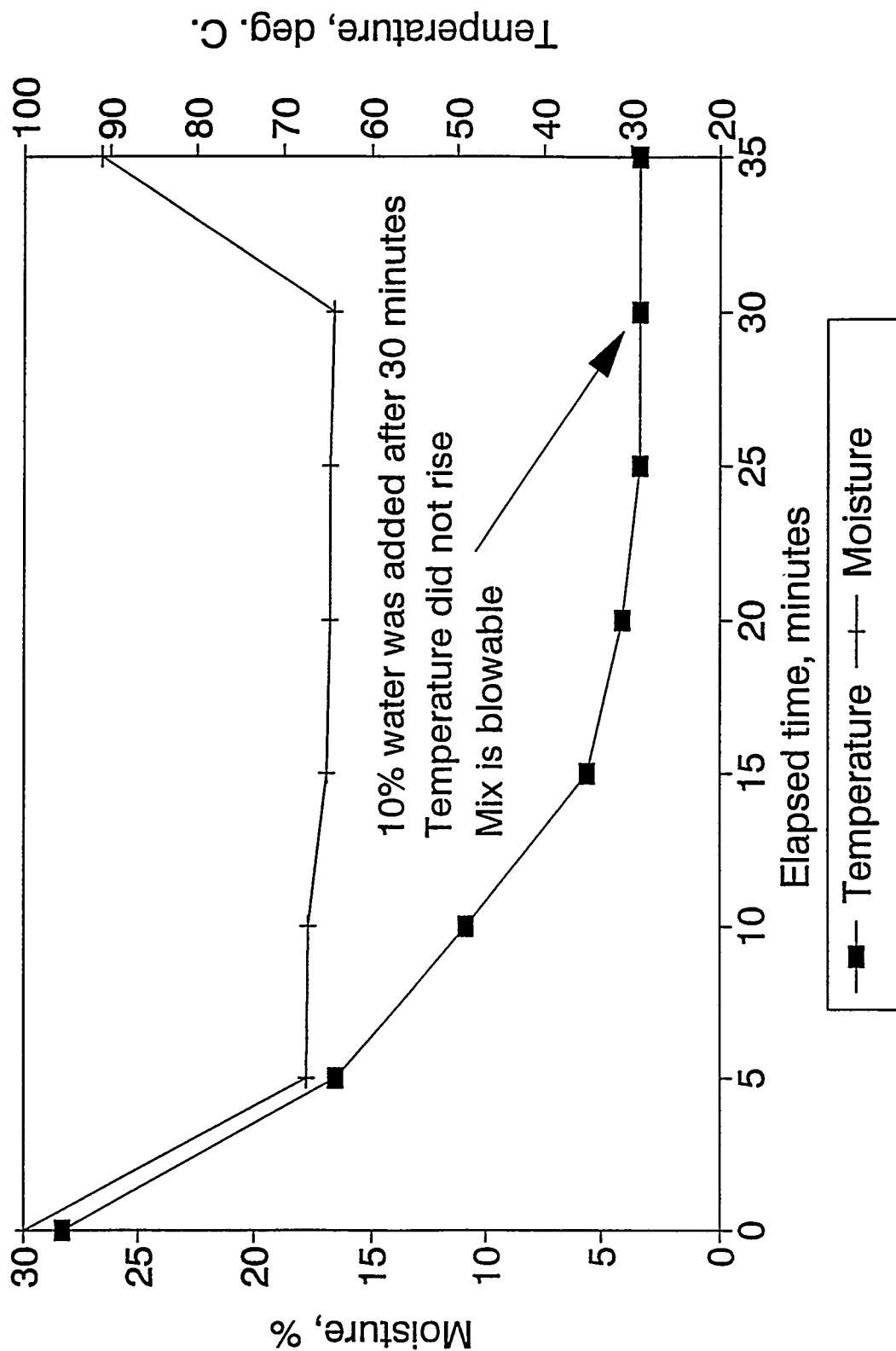


Figure 11 Temperature vs. elapsed time when simulating a drum type mixer

Stress-strain curve (7-day test)

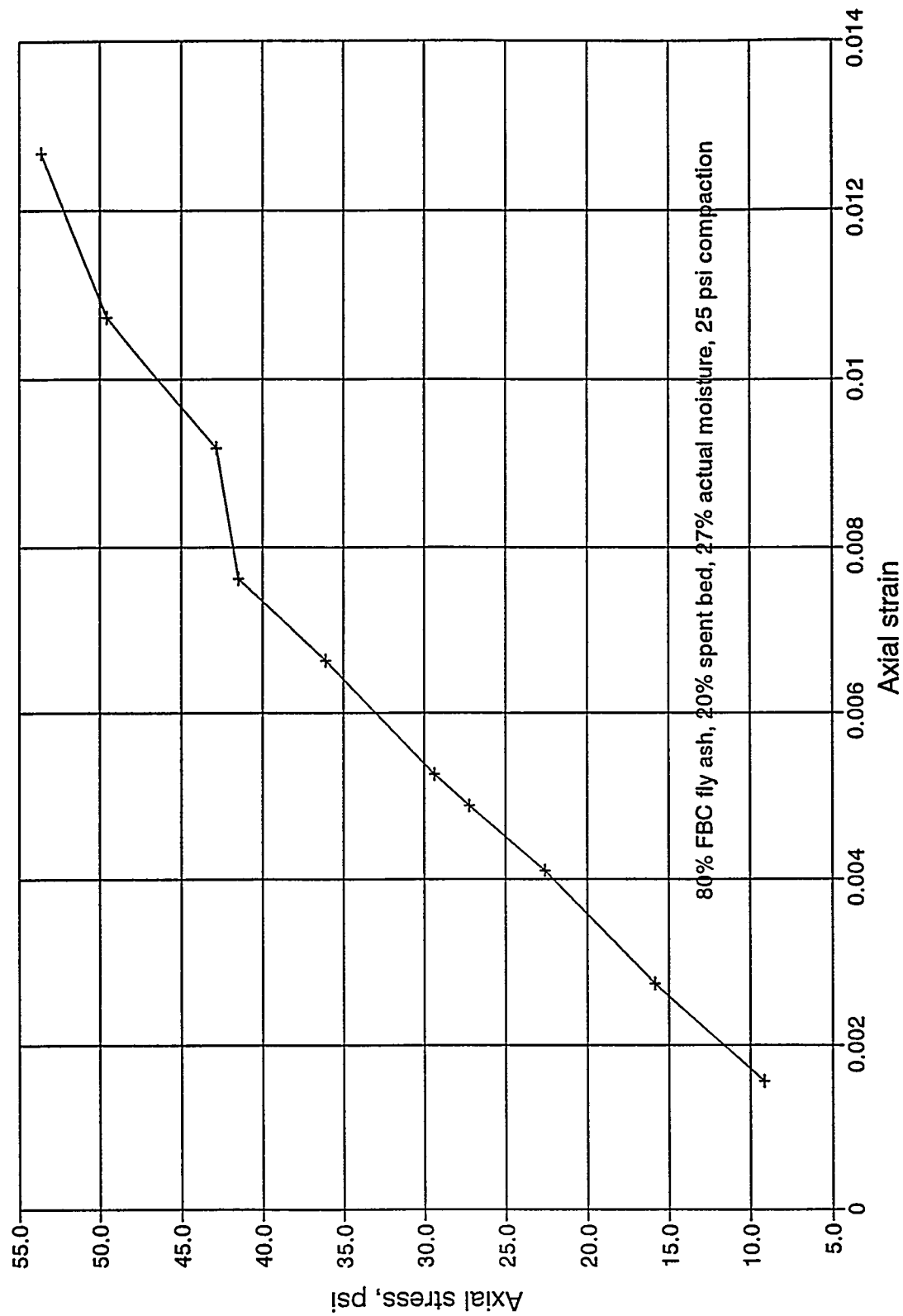


Figure 12 Typical stress-strain curve for 7-day test

Stress-strain curve (90-day test)

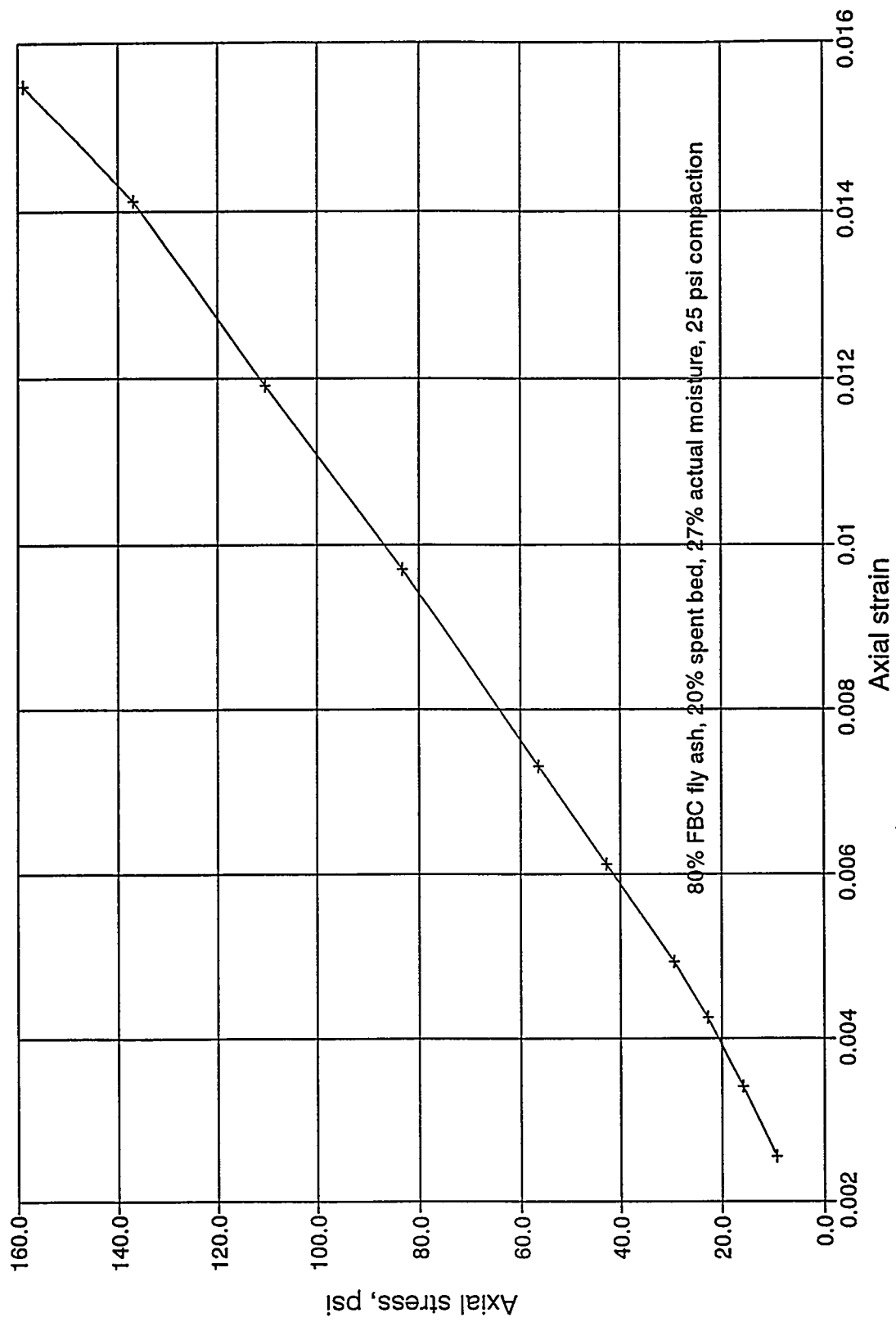


Figure 13 Typical stress-strain curve for 90-day test

MANAGEMENT OF DRY FLUE GAS DESULFURIZATION BY-PRODUCTS IN UNDERGROUND MINES

Technical Progress Report -- April 1 - June 30, 1995

MIX CHARACTERIZATION, NUMERICAL MODELING, AND FIELD INSTRUMENTATION

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Chugh, Y. P., J. Daugherty, et al. (1989) Laboratory characterization of weak floor strata associated with coal seams in Illinois, *Final Report to Illinois Mine Subsidence Research program*, Champaign, IL.

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SECTION VI

HYDRAULIC PLACEMENT

DR. X. YUAN
CO-PRINCIPAL INVESTIGATOR

MANAGEMENT OF DRY FLUE GAS DESULFURIZATION BY-PRODUCTS IN UNDERGROUND MINES

Technical Progress Report -- April 1 - June 30, 1995

HYDRAULIC PLACEMENT

RECENT ADVANCES IN THIS RESEARCH PROGRAM

During this quarter, tests on the slump characteristics and mechanical properties of the hydraulic mixes--the mixtures of fly ash, scrubber sludge, lime waste, and water, - have been conducted.

1). SLUMP TEST

For the determination of the slump characteristics, the moisture content in the mixtures of fly ash, scrubber sludge, and lime waste, were considered. Two different sizes of test cones were used so as to determine the effect of cone size on slump height. The larger cone is 12 inches high, with 3 inch top diameter, and 6 inch bottom diameter (Figure 1 (a)). The small one is 6 inches high, with 1.5 inch top diameter and 3 inch bottom diameter (Figure 1 (b)).

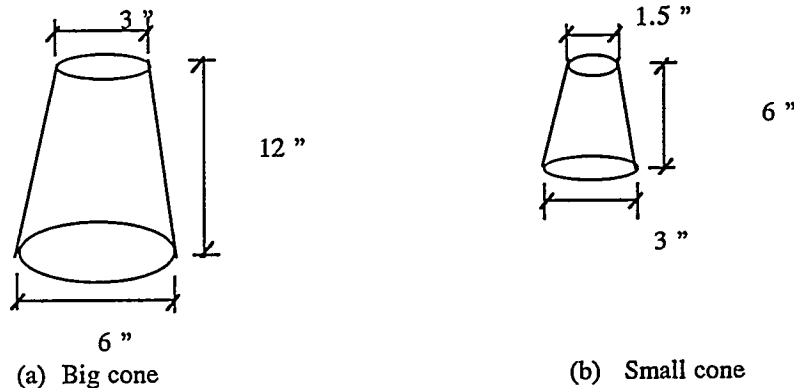


Figure 1 Cones for slump test

The mixtures include the following situations:

- (1). 40% fly ash, 55% scrubber sludge, 5% lime waste, 28% water;
- (2). 40% fly ash, 55% scrubber sludge, 5% lime waste, 28.5% water;
- (3). 40% fly ash, 55% scrubber sludge, 5% lime waste, 29% water;
- (4). 40% fly ash, 55% scrubber sludge, 5% lime waste, 30% water.

MANAGEMENT OF DRY FLUE GAS DESULFURIZATION BY-PRODUCTS IN UNDERGROUND MINES

Technical Progress Report -- April 1 - June 30, 1995

HYDRAULIC PLACEMENT (CONTINUED)

A. Slump height

The slump heights are shown in Figure 2.

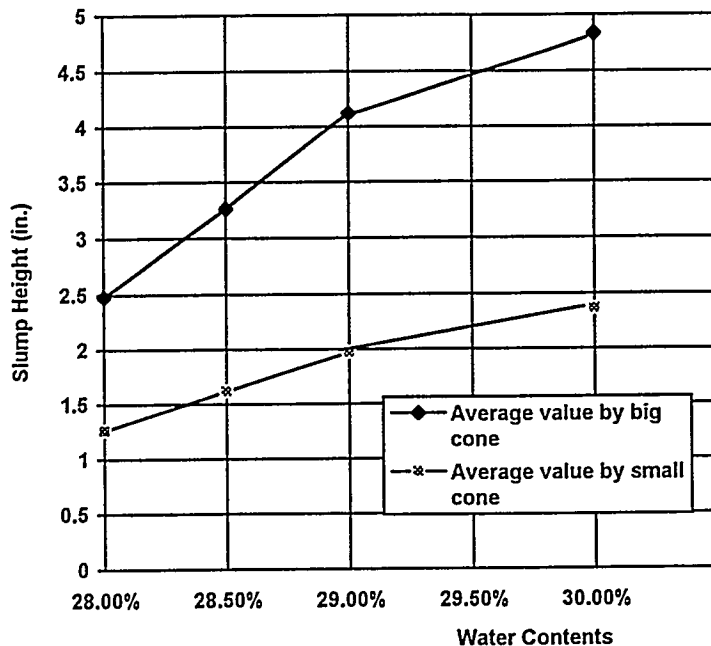


Figure 2 Curve of slump heights Vs moisture contents

The relationship between the slump heights by the big cone and those by the small cone is plotted in Figure 3, from which it can be seen that the slump height by the small cone is roughly half of that by the larger cone.

MANAGEMENT OF DRY FLUE GAS DESULFURIZATION BY-PRODUCTS IN UNDERGROUND MINES

Technical Progress Report -- April 1 - June 30, 1995

HYDRAULIC PLACEMENT (CONTINUED)

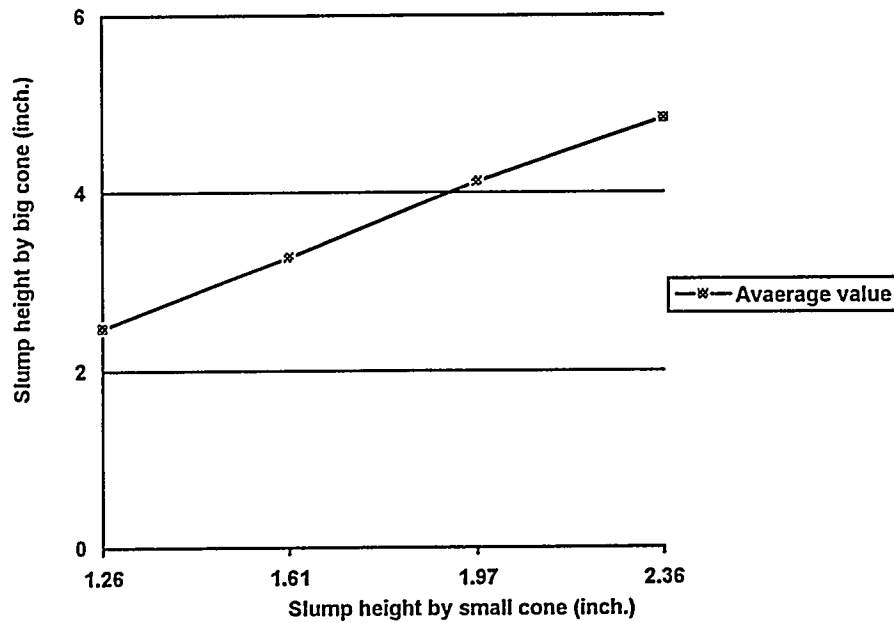


Figure 3 Relationship between the slump heights by the larger cone and those by the small cone.

B. Power used in each slump test

In each slump test, the power used in the mixing of hydraulic materials was measured. Figure 4 gives the results.

MANAGEMENT OF DRY FLUE GAS DESULFURIZATION BY-PRODUCTS IN UNDERGROUND MINES

Technical Progress Report -- April 1 - June 30, 1995

HYDRAULIC PLACEMENT (CONTINUED)

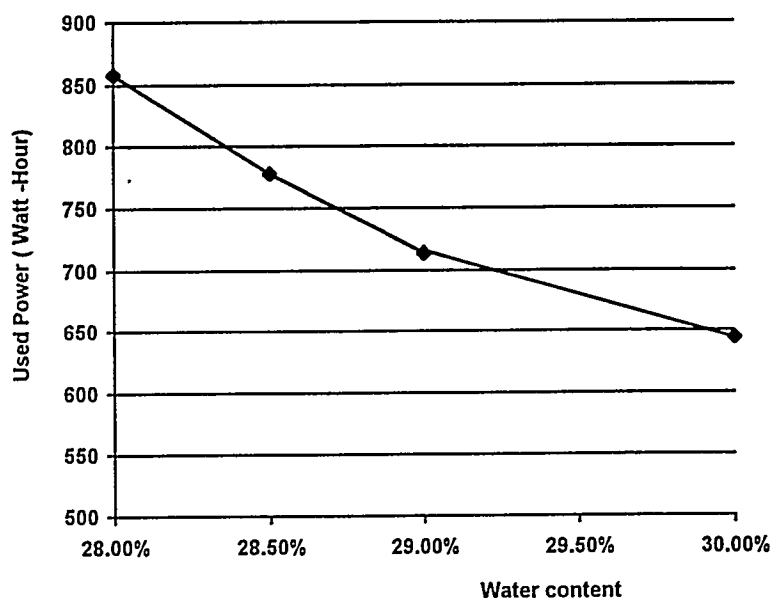


Figure 4 Relationship of used power with moisture content

It can be seen that the power required for the mixing of materials will decrease with the increase of the moisture content.

2). STRENGTH TEST

To determine the strength characters of hydraulic materials with different curing times, two groups of samples were tested.

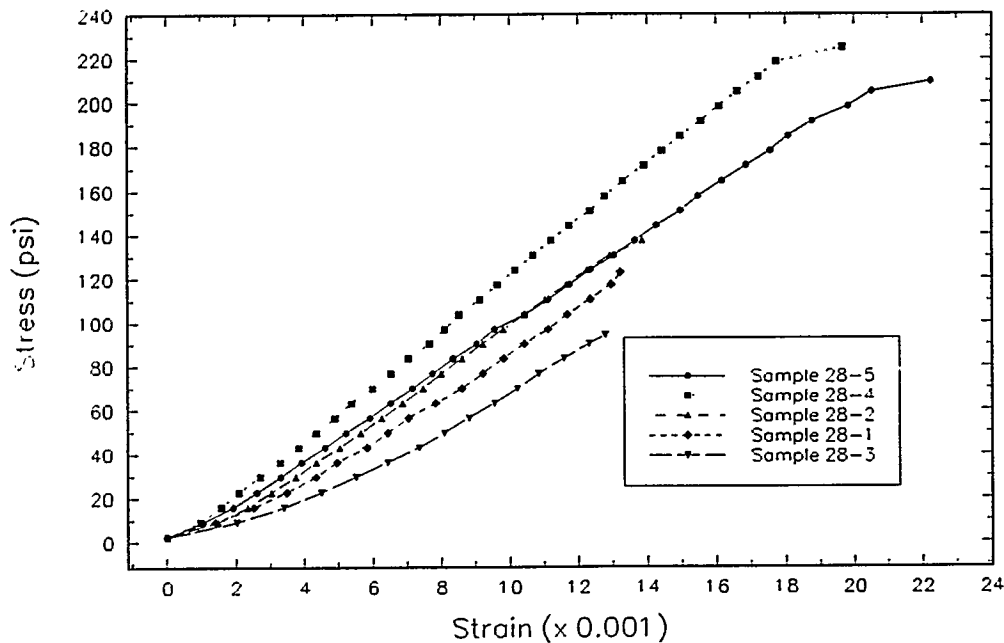
Group 1 consists of 5 cylindrical samples, composed of 40% flay ash, 55% scrubber sludge, 5% lime waste, and 31% water. After molding the samples were put into an aerial curing chamber. Twenty-five days later these samples were tested. The average axial compressive strength was 157.71 psi, and the average Young's modulus was about 9900 psi. Figure 5 shows the test results.

MANAGEMENT OF DRY FLUE GAS DESULFURIZATION BY-PRODUCTS IN UNDERGROUND MINES

Technical Progress Report -- April 1 - June 30, 1995

HYDRAULIC PLACEMENT (CONTINUED)

Fig.5 Relation between stresses and strains of sample group 1
(40% F.A; 55% S.S; 5% L.W; 31% W.C, curing 25 days)



Group 2 includes 4 cylinder samples, which were made of 40% fly ash, 55% scrubber sludge, 5% lime waste, and 29% water. The curing time for this group of samples is 35 days. The average axial compressive strength is about 184.221 psi, and the corresponding Young's modulus is about 10100 psi. Figure 6 shows the test results of this group.

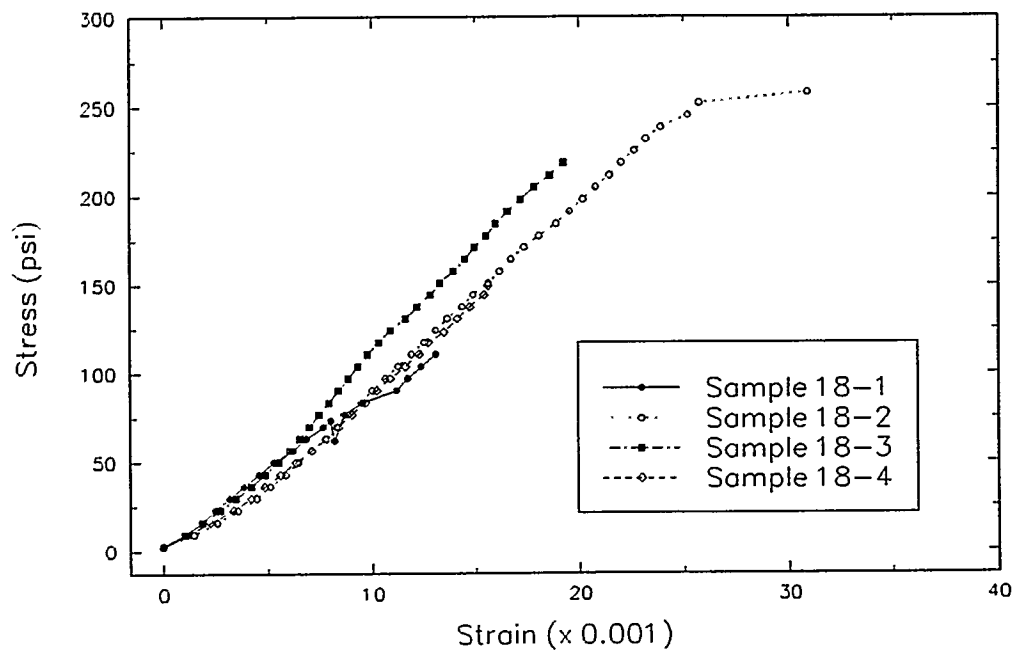
Compared with the strength test results reported in the last quarterly report, it can be seen that the strength of the samples which were cured in a longer time period will be greater than that of the samples which were cured in a shorter time period. Figure 7 and Figure 8 give these comparison results.

MANAGEMENT OF DRY FLUE GAS DESULFURIZATION BY-PRODUCTS IN UNDERGROUND MINES

Technical Progress Report -- April 1 - June 30, 1995

HYDRAULIC PLACEMENT (CONTINUED)

Fig. 6 Relation between stresses and strains of sample group 2
(40% F.A; 55% S.S; 5% L.W; 29% W.C; curing 35 days)



MANAGEMENT OF DRY FLUE GAS DESULFURIZATION BY-PRODUCTS IN UNDERGROUND MINES

Technical Progress Report -- April 1 - June 30, 1995

HYDRAULIC PLACEMENT (CONTINUED)

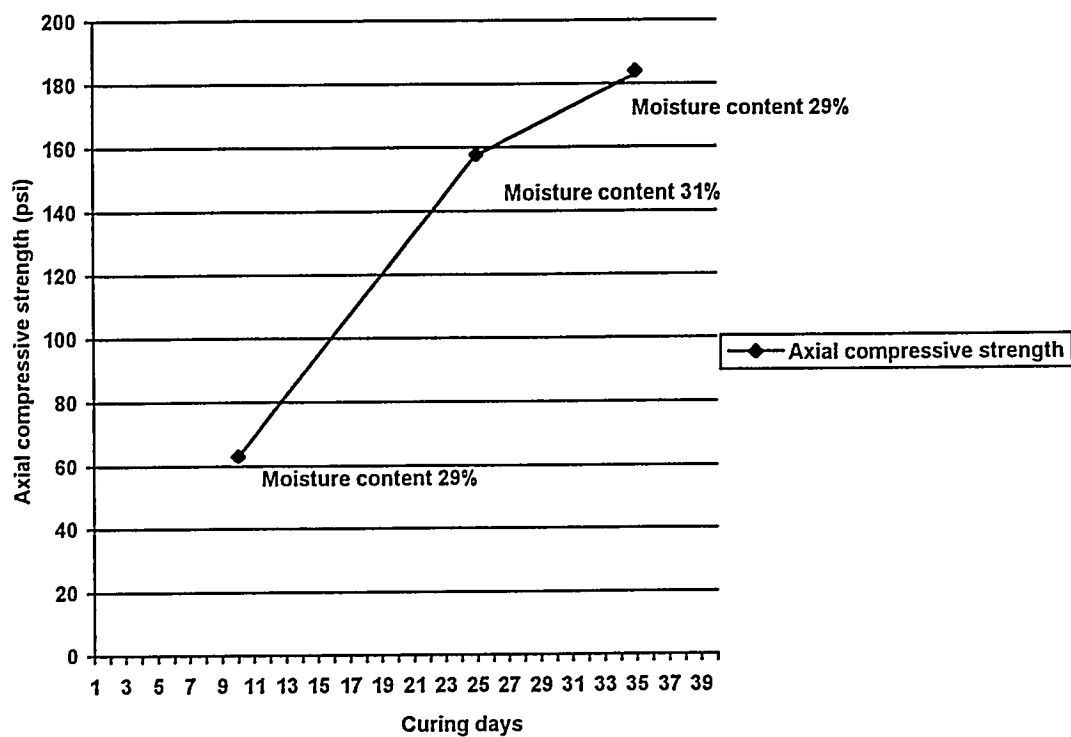


Figure 7. Relation of strength and curing time

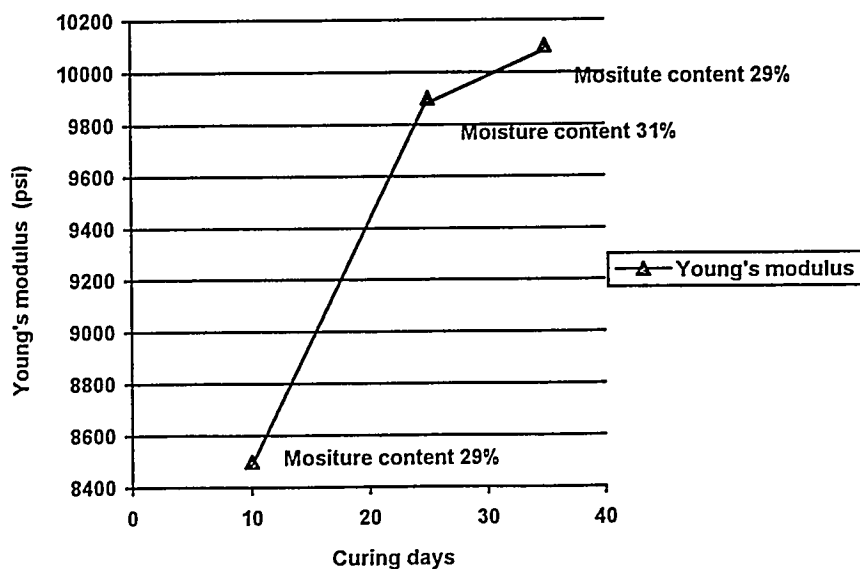


Figure 8 Relation of Young's modulus and curing time

MANAGEMENT OF DRY FLUE GAS DESULFURIZATION BY-PRODUCTS IN UNDERGROUND MINES

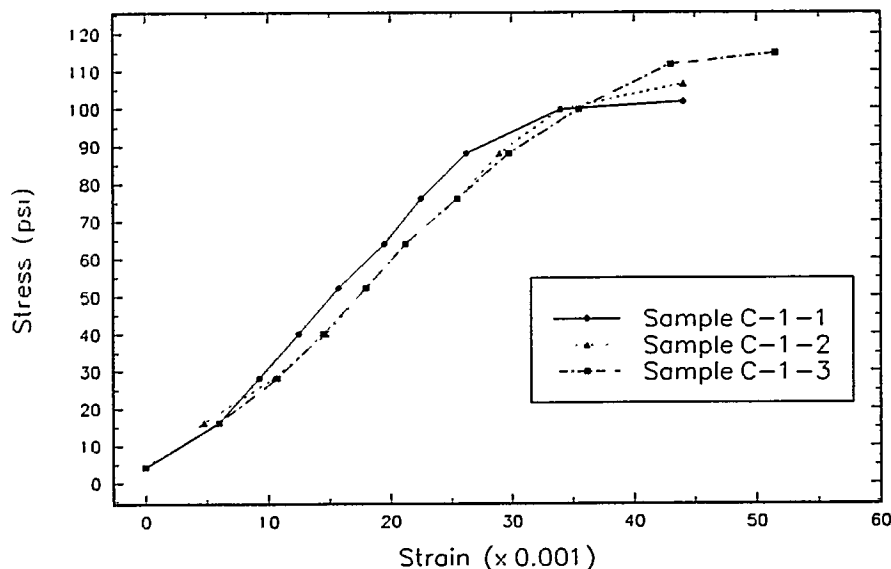
Technical Progress Report -- April 1 - June 30, 1995

HYDRAULIC PLACEMENT (CONTINUED)

Additionally, for the investigation of the influence of lime waste content on the strength of hydraulic materials, two groups of cubic samples were tested. The size of each sample is 2" x 2" x 2". The first group includes samples C-1-1, C-1-2, and C-1-3, which were made of 38% fly ash, 52% scrubber sludge, 10% lime waste, and 33% water. The second group consists of samples C-2-1 and C-2-2, which were made of 35% fly ash, 50% scrubber sludge, 15% lime waste, and 31% water. After being shaped and kept at room temperature for one day, both groups of samples were put into a steam chamber where the temperature is kept around 80 centigrade. After 24 hours' curing in this steam chamber, the samples were tested. For the first group of samples, the average compressive strength and Young's modulus are separately 106.6 psi and 3,000 psi; and for the second group, separately 163.25 psi and 3,600 psi. Figure 9 and Figure 10 show the corresponding results.

From these strength test results, it can be seen that, with the increase of lime waste, the strength of hydraulic materials will be increased.

Figure 9. Relation of stresses and strains
(38% F.A, 52% S.S, 10% L.W, 33% W.C, One day's steam chamber curing)

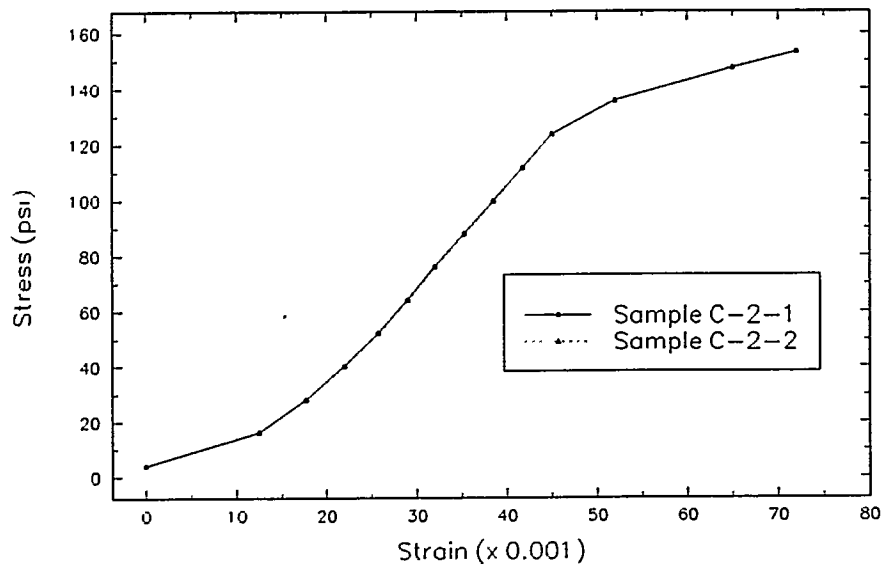


MANAGEMENT OF DRY FLUE GAS DESULFURIZATION BY-PRODUCTS IN UNDERGROUND MINES

Technical Progress Report -- April 1 - June 30, 1995

HYDRAULIC PLACEMENT (CONTINUED)

Figure 10. Relation of stresses and strains
(35% F.A, 50% S.S, 15% L.W, 31% W.C, One day's steam chamber curing)



SECTION VII

THE NEXT QUARTER

MANAGEMENT OF DRY FLUE GAS DESULFURIZATION BY-PRODUCTS IN UNDERGROUND MINES

Technical Progress Report -- April 1 - June 30, 1995

THE NEXT QUARTER

A major highlight in the next quarter will be the annual Department of Energy oversight review of the three ongoing coal combustion residues management programs at the University of Kentucky, Southern Illinois University at Carbondale, and the West Virginia University. The workshop/meeting will be held September 26 and 27 at the SIUC Student Center.

A concentrated effort to develop the pneumatic and hydraulic injection systems will be undertaken in cooperation with Mine Systems Design and Eric Powell Associates. The objective will be to design and specify the hardware needed to accomplish the injections at the planned rates. Hardware acquisition will not be started during the quarter, however.

Monitoring wells at the Peabody mine #10 injection site are planned to be drilled during this quarter, and subsidence monuments replacing those destroyed will be installed. Likewise, work will continue on the engineering and economic evaluation model.

The next quarter will mark the end of two years of the program, and a second annual report will be prepared detailing the progress of the program from its inception.

APPENDIX I

PAPER TO BE PRESENTED AT
THE PITTSBURGH COAL SHOW

ECONOMICS OF COAL COMBUSTION RESIDUE TRANSPORTATION

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ABSTRACT

A group of researchers at the Southern Illinois University is engaged in a research project whereby technical, environmental, and economic feasibility of coal combustion residue disposal into old underground coal mines is being investigated. Safe and economic transportation of residues from power plants to mine sites is an important segment of this project. A number of transportation alternatives have been examined, and among these, pneumatic trucks, pressure differential rail cars, and collapsible intermodal containers have been found to be promising. In this paper, all three alternatives are applied to hypothetical cases pertaining to central and southern Illinois. The operating scenarios are described and a comparative economic analysis is conducted using "After-Tax Cost" method. Each alternative is evaluated for varying distances and tonnages to reveal its favorable operating range.

INTRODUCTION AND BACKGROUND

The disposal of coal combustion residues in an environmentally sound and economically acceptable manner is a growing concern in the coal and utility industries in the US. In 1990, approximately 100 million tons of combustion residues were produced on a dry basis. The residue production is expected to increase significantly after 1995 due to the Clean Air Act Amendment of 1990. To meet the requirements of the Act, numerous power plants will have to adopt advanced combustion and flue gas desulfurization (FGD) technologies which will significantly increase the amount of by-products. The current most widely used residue disposal method is surface disposal to landfills or ponds near the power plants. Surface disposal has a number of environmental concerns such as surface and subsurface acidic discharge, dust, decreased land value and land use. Also, surface disposal is becoming costlier due to increasing residue production, stringent environmental regulations, and diminishing land availability.

Underground disposal of residues may alleviate the problems associated with surface disposal. Furthermore, since the FGD by-products typically have high calcium hydroxide content and cementitious properties, if properly disposed in the old workings of the underground coal mines they may mitigate subsidence and acid mine drainage.

Under a contract from the US Department of Energy, Southern Illinois University at Carbondale (SIUC) is investigating the technical, environmental, and economic feasibility of residue disposal into the old workings of the coal mines. The project is focused on Illinois and it is progressing successfully in 5 coordinated areas since September 1993. These areas are: 1) Residue Characterization, 2) Materials Handling, 3) Underground Residue Placement, 4) Field Demonstrations, and 5) Economic Evaluation.

In this paper, the progress in materials handling and systems economics areas is reported. The objective of materials handling research is to identify the systems that are technically, economically, and environmentally feasible in handling and transporting the coal combustion residues from the power plant to the injection site. To achieve this objective a number of alternatives have been evaluated, and few more are still under investigation. So far, the research has been focused on dry residue transportation. Among the alternatives evaluated, three were found to be promising; pneumatic trucks, pressure differential rail cars, and collapsible intermodal containers (CICTM).

Pneumatic Trucks (PT)

Pneumatic trucks, also referred to as bulk tank trucks, are widely used in transporting low density dry flowable powder and granules as well as high density materials such as cement, limestone and fly ash. A pneumatic truck is composed of three main components: (i) tractor, (ii) tank trailer, and (iii) blower. The most common method of loading the material is gravity feeding from a silo by a collapsible spout which engages to the gate on top of the tank. The tank is air tight when the lids of the gates are closed. During offloading, the material flows through the piping below the tank due to pressure difference created by the blower. The PT transportation scenario for this project is schematically shown in Figure 1. The trucks are loaded from the fly ash bin of the plant and they deliver the material directly to the injection point at the mine site. There, the pressure necessary for offloading the fly ash into the injection hopper is supplied by the blower mounted on the truck. These trucks are approximately 25 tons in capacity and can offload in about 25 minutes.

Pressure Differential Rail Cars (PD cars)

These are special type of rail cars used to handle powdered materials. They are operated under the principle of pressure differences between the car and the container to which the product is discharged. Normally, PD cars are complemented with pneumatic trucks at rail terminals to deliver the material to the final destination. When a PD car is pressurized to about 5 psi or more, the outlet valves are opened to form a steady flow of material into the truck until all the material in the compartment is cleared out.

The PD car transportation scenario is schematically shown in Figure 2. As seen in this figure, there are three sets of rail cars; the first set of cars is at the plant and is being filled from the fly ash bins; the second set of cars, which has already been filled and attached to the empty coal unit train, is traveling to the mine; and the third set is at the mine site and is being offloaded into pneumatic trucks at a pace synchronized with the injection system capacity.

When the train arrives at the mine, the PD cars are demurred and parked at the junction of the siding. Then, the unit train pulls under the coal silo for coal loading. After coal loading is completed, the empty PD cars which have been waiting at the siding are attached at the back of the loaded unit train. The train then leaves for the plant. Meanwhile, the loaded PD cars which have been left at the junction are pulled into the rail siding and their content is transferred into a silo by the use of a blower. The pneumatic trucks fill their tanks from this silo and then deliver the material to the injection hopper.

When the train arrives at the plant, the empty PD cars are demurred and parked at the junction of the siding. The unit train pulls over the undertrack bin and dumps the coal. After coal dumping is completed, the PD cars which have already been filled and waiting for the train are attached to the back of the empty coal train. Subsequently, the train leaves for the mine. Meanwhile, the empty PD cars are pulled into the rail siding and the fly ash loading process restarts.

The above described operating scenario is valid only when the tonnages of coal and fly ash transported justify a dedicated unit train between the plant and the mine. Otherwise, two sets of PD cars will suffice. The first set will be at the plant site, and when it is filled, a local train will deliver them to the mine. Meanwhile, the empty PDs will be brought back from the mine by the coal train. The exchange of empty cars with the full ones will be scheduled in such a way that there will be no excess accumulation of residue at the mine site.

Collapsible Intermodal Containers (CICTM)

These containers are made of rubber coated aramid and nylon fabric with polyester webbing. They are patented by SEECTM Inc., one of the research partners. The CICs are collapsible storage bins that are portable and intermodal - designed to ride inside coal cars, barges and trucks. Those CICs made to transport fly ash by riding in coal cars have a height of 120 inches, diameter of 110 inches and a 19-inch filling port. For ash of 60 lb. per cu. ft. bulk density, the CIC capacity is about 20 tons. These containers are extremely durable and provide fully encapsulated transport, eliminating fugitive dust problems.

The CIC transportation scenario is shown schematically in Figure 3. The coal train arrives at the plant and offloads coal into an undertrack bin. Next, the CICs which have already been filled with fly ash and staged along the rail are lifted, one at a time, by an overhead crane and placed into the bays of the empty coal cars. Two specially designed lifting brackets mounted on both sides of the CIC facilitate lifting and placement by the crane into the bay of the car. Four CICs occupy a car, each taking one of the four bays of a typical coal car. The overhead crane is on rubber tires and travels along the rail track looking inside the cars. When all the CICs are loaded, the train leaves for the mine.

At the mine site, an overhead crane lifts the CICs, one at a time, and places them on the concrete pad along the rail track. When all the CICs are offloaded, the train pulls under the silo for coal loading. After filling all the hopper cars, the train leaves for the power plant. The CICs are then loaded on tote trailer(s) by the same crane and transported to the injection site. There, the ash is offloaded into the hopper of the injection system by the use of a

vacuum system designed for the CICs. The empty CIC can be transported back to the rail site on the same trailer. At the rail site, the empty bag is lifted with a small fork lift, carried into a baghouse where the air trapped in the CIC is extracted. The collapsed bag is then retrieved by the forklift and hung like a vest onto the rail guides of a covered trailer. After collecting 25-40 empty CICs, the trailer is transported back to the plant.

At the plant, the tractor leaves the filled trailer, takes the empty trailer and drives back to the mine. The empty CICs are retrieved from the trailer with the help of a small forklift and placed on a specially designed trailer, one at a time, and pulled under the fly ash silo by a tractor. There, it is filled by gravity similar to filling a pneumatic truck. The CIC then is transported back to the rail site where the trailer pulls under the overhead crane and the CIC is lifted and staged along the track and kept there until the coal train comes back from the mine.

ECONOMIC EVALUATION

The alternatives were applied to the conditions of central and southern Illinois. Engineering analyses were conducted for each alternative to determine the values of the system parameters and the operating and capital costs. The systems then were evaluated using the "Net Cash Cost (NCC)" method which is also known as the "After-Tax Cost" method. For each year of the project life, the NCC is calculated using the following equation:

$$\text{NCC} = (\text{Operating Cost} + \text{Depreciation}) (1 - \text{tax rate}) - \text{Depreciation} + \text{Capital Cost} \quad (1)$$

The alternatives were evaluated at three levels of transportation distance; 30, 100, and 200 miles, and at three levels of production rates; 100,000, 200,000, and 300,000 tons per year, to determine the range in which they are favorable. It is noted that in central and southern Illinois, most of the underground mines where residues can be disposed are not more than 200 miles away from the power plants, and the annual residue production does not exceed 300,000 tons. It is assumed that the mine sells 3 tons of coal to the plant for every ton of residue it receives from that plant.

The projects were evaluated over five years, at the end of which all capital cost items were liquidated at fair market prices. A minimum required rate of return of 12 %, an effective tax rate of 40 %, and MACRS depreciation method were applied to all projects. All cost estimates were made in terms of 1995 dollars, and the rate of cost increases is assumed to be the same as the inflation rate so that the evaluations can be done in constant dollars. Some of the important cost data and assumptions specific to each alternative is given below.

Pneumatic Trucks: The depreciation and useful lives of pneumatic trucks are 5 and 10 years, respectively. The capital cost of a unit is approximately \$120,000, and the salvage value of a 5-year old unit is assumed to be 50% of its original cost. The number of units in the fleet is determined in such a way that the fleet availability is at least 80% at all mines.

PD Cars: The depreciation and useful lives of PD cars are 7 and 25 years, respectively. The capital cost of a PD car is approximately \$80,000, and the salvage value of a 5-year old car is

assumed to be 80% of its original cost. This system requires 1/4 mile rail siding both at the plant and the mine site. The rail company will charge the "multiple-car" rate which will be 4 to 5 times higher than the "backhaul" rate since this system will not fit in "backhaul" classification. A car will only qualify for backhaul rate if it is utilized to transport residue when it goes back to the origin.

CICs: The depreciation and the useful lives of CICs are 5 and 20 years, respectively. The capital cost of a CIC is \$15,000, and the salvage value of a 5-year old CIC is assumed to be 75% of its original cost. This system requires a 50-foot wide by 1/4 mile long concrete pad along the rail track at both the plant and the mine sites to accommodate the overhead cranes. At the rate of 300,000 tons per year, this system will be charged with a "backhaul" rate by the railroad. However, for lesser tonnages it will be treated as a "multiple-car" since a dedicated unit train can not be justified.

INTERPRETATION OF THE RESULTS

For each particular alternative, a price-per-ton value is calculated for each distance-tonnage combination. Table 1 illustrates the calculation of the price-per-ton when 100,000 tons of ash is transported for 30 miles by PTs. As seen, \$360,000 were invested at time 0 to purchase 3 trucks to do the job. The negative cash flows in the next five years indicate that these are costs in each year. The salvage value of these trucks at the end of five years is \$180,000, bringing an after-tax capital gain of \$116,000. The annual equivalent cost is calculated to be \$312,000 at 12% discount rate. The price to be charged per ton of material transported is \$5.21. This value is obtained by dividing the after-tax cost of \$3.12/ton by (1- tax rate). In other words, if \$5.21 is plugged back into the economic model, the NPV of the project would be exactly 0, indicating a return of 12% on investment. The outputs of the 27 runs are presented in Figure 4. The following observations can be made from Figure 1:

1. The price to be charged in PT alternative is not sensitive to tonnage, but very sensitive to distance. That is, the longer the distance the higher the price.
2. The prices to be charged in PD-car and CIC alternatives are not sensitive to distance, but very sensitive to tonnage. That is, the more the tonnage the lower the price.
3. For 100,000-ton annual production, PT alternative provides prices that are lower than those of PD-car and CIC alternatives up to a distance of approximately 150 miles. For shorter distances, the difference in prices between the PT and the other two alternatives is very significant. The prices of PD-car and CIC alternatives show similar trend, but on the average, the price of PD-car runs \$0.9 less than the CIC.
4. For 200,000-ton annual production, PT alternative provides prices lower than those of the PD-car and CIC alternatives up to a distance of approximately 90 miles. Again, the difference is very significant at shorter distances. The prices of PD-car and CIC show again similar trend, but on the average, the price of PD-car runs \$0.7 less than the CIC.

5. For 300,000-ton annual production, PT alternative is lower priced than PD-car and CICs up to 60 miles in distance. Otherwise, the CIC price is on the average \$1.80 lower than the PD-car, reversing the price differential seen between the PD-car and CICs at lower tonnages. This is due to the CICs being charged at a rail backhaul rate at 300,000 tons per year.

CONCLUSIONS

The results above are based on cost estimates and system assumptions for central and southern Illinois. Importantly, this model can be applied elsewhere by substituting the actual costs and system factors for other plant sites and geographic regions. As seen in item 5 above, price sensitivity is such that a backhaul rate at any tonnage can reverse the economics in favor of CICs, especially for long distances. In a final system decision, one must also consider intangibles. For example, in moving large tonnages, the impact of increased truck traffic and road deterioration may favor CICs. It is also noted that the transportation option between a power plant and a mine may be narrowed by existing facilities at these sites. For instance, availability of rail sidings may favor PD cars over other alternatives.

ACKNOWLEDGMENT

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Table 1. Calculation of Transportation Cost for Pneumatic Trucks
(30 Miles - 100,000 Tons Case - central and southern Illinois)

Year	0	1	2	3	4	5
Revenue.....	0.	0.	0.	0.	0.	0.
-Operating Cost.	0.	-432.	-432.	-432.	-432.	-432.
-Depreciation...	0.	-72.	-115.	-69.	-41.	-41.
Taxable Income..	0.	-504.	-547.	-501.	-473.	-473.
-Tax Savings....	0.	202.	219.	200.	189.	189.
Net Income.....	0.	-302.	-328.	-301.	-284.	-284.
+Depreciation...	0.	72.	115.	69.	41.	41.
-Capital Cost...	-360.	0.	0.	0.	0.	0.
Net Cash Flow...	-360.	-230.	-213.	-232.	-243.	-243.
AFTER-TAX CAPITAL GAIN (\$1000)						= +116.
NET PRESENT VALUE (\$1000).....						= -1126.
AFTER-TAX ANNUAL EQUIVALENT COST (\$1000) ...						= -312.
AFTER-TAX COST PER TON (\$)						= -3.12
BEFORE-TAX PRICE (\$)						= 5.21

APPENDIX II

PACKER TESTS; CRITICAL LITERATURE REVIEW

Packer Tests: Critical Literature Review

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1. INTRODUCTION

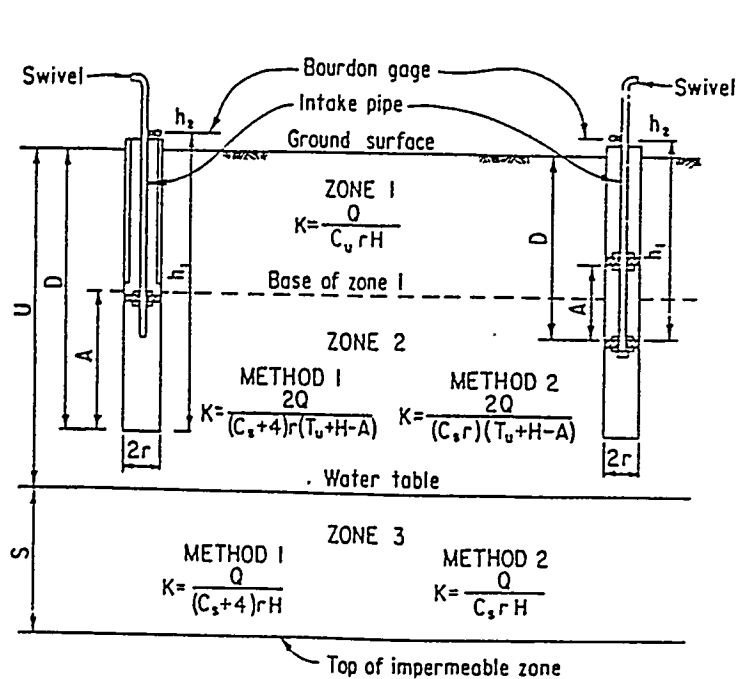
1.1 Objectives

Packer tests are used to estimate the hydraulic conductivity of a geologic formation. The purpose of this work is to provide and present a set of procedures to obtain and mathematical computations used to analyze data from a packer test. Examples dealing with single and double packer tests are analyzed using the equations and procedures presented in the report. Finally, some conclusions and observations about packer testing and analysis are presented.

1.2 Conventions and variables used in this report

The conventions and variables used in this report are consistent with those used in the Ground Water Manual [USBR, 1985]. The following variables were used (most are shown on figure 1):

- A = length of test section [L].
- a = surface area of test section [L²].
- C_s = conductivity coefficient for semispherical flow in saturated materials through partially penetrating cylindrical test wells [dimensionless].
- C_u = conductivity coefficient for unsaturated material with partially penetrating cylindrical test wells [dimensionless].
- D = distance from ground surface to bottom of test section [L].
- H = h₁ + h₂ - L = effective head [L].
- h₁ (above water table) = distance between Bourdon gage and bottom of hole for single packer or distance between gage and upper surface of lower packer for double packer [L].
- h₁ (below water table) = distance between gage and water table [L].
- h₂ = applied pressure at gage [L].
- K = coefficient of permeability [LT⁻¹].
- L = head loss in pipe due to friction [L]; ignore head loss for Q < 4 gallons per second in a 1.25" pipe; use length of pipe between gage and top of test section for computations.
- Q = steady flow into the well [L³T⁻¹].
- r = radius of test hole [L].
- S = thickness of saturated material [L].
- U = thickness of unsaturated material [L].
- T_u = U - D + H = distance from water surface in well to water table [L].
- X = 100 H / T_u = percent of unsaturated stratum [dimensionless].



- K = coefficient of permeability, feet per second under a unit gradient
 Q = steady flow into well, ft³/s
 $H = h_1 + h_2 - L$ = effective head, ft
 h_1 (above water table) = distance between Bourdon gage and bottom of hole for method 1 or distance between gage and upper surface of lower packer for method 2, ft
 h_1 (below water table) = distance between gage and water table, ft
 h_2 = applied pressure at gage, 1 lb/in² = 2.307 ft of water
 L = head loss in pipe due to friction, ft; ignore head loss for $Q < 4$ gal/min in 1 1/4-inch pipe; use length of pipe between gage and top of test section for computations
 $X = \frac{H}{U} (100)$ = percent of unsaturated stratum
 A = length of test section, ft
 r = radius of test hole, ft
 C_u = conductivity coefficient for unsaturated materials with partially penetrating cylindrical test wells
 C_s = conductivity coefficient for semi-spherical flow in saturated materials through partially penetrating cylindrical test wells
 U = thickness of unsaturated material, ft
 S = thickness of saturated material, ft
 $T_u = U - D + H$ = distance from water surface in well to water table, ft
 D = distance from ground surface to bottom of test section, ft
 a = surface area of test section, ft²; area of wall plus area of bottom for method 1; area of wall for method 2
 Limitations:
 $Q/a \leq 0.10$, $S \geq 5A$, $A \geq 10r$, thickness of each packer must be $\geq 10r$ in method 2

Figure 1. System configuration and variables used (from USBR, [1985])

2. DESCRIPTION OF PACKER TESTS

2.1 General background on packer tests

Packer tests are pressure tests in which a one or two packers are used to isolate a part of a borehole into rock. The borehole does not need to be vertical, it can be angled or even horizontal (Ground Water Manual [USBR, 1985]). Water is injected at a known pressure and, at steady state conditions, the water discharge is measured, and with that information the value of the conductivity can be estimated. According to Braester and Thunvik [1985], packer tests are used for in-situ determination of the hydraulic conductivity of a geological formation, especially in formations with low values of hydraulic conductivity such as hard rock. These methods were originally developed for the exploration of prospective sites for conventional structures like dams and tunnels; it has been used recently mainly for the investigation of sites for radioactive waste repositories and underground hydrocarbon storage reservoirs

2.2 Single packer tests

Single packer tests are used when determining the hydraulic conductivity of a consolidated rock formation that would require the installation of casing during drilling. However, the hole should be neither cased nor cemented at the moment the packer test, otherwise the results would be wrong. In general the procedure to follow is:

- The hole is drilled to the maximum depth so that there is no need to immediately case it.
- The drilling tools are then removed.
- The hole should be washed and bailed or blown out thoroughly in order to minimize "borehole skin effects". A detailed discussion on these effects is presented in section 2.5 of this report.
- The packer is seated above the bottom of the drilled hole so that the test section is included completely in the "just-drilled" portion of the hole. Care must be taken with the packer so that "equipment compliance effects" are minimized as much as possible. These effects are presented in detail in section 2.4 of this report.
- Water is then pumped until steady state is reached (i.e., three or more consecutive readings of pressure and discharge taken at five minute intervals are approximately constant). It is important to get steady state conditions since compliance effects are minimized this way. Pickens et al. [1987] noted that the most important compliance effect is observed at shut in.
- Pressure and discharge readings are recorded at 5 minute intervals.
- The packer is removed. The test can be repeated in other formations as the hole is drilled.

2.3 Double packer tests

Double packer tests are used when determining the hydraulic conductivity of a consolidated rock formation that is stable and does not require either casing or cementing during drilling. In general, the procedure to follow is:

- The hole is drilled to the desired depth.

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- The drilling tools are then removed.
- The hole should be cleaned and bailed or blown out thoroughly in order to minimize "borehole skin effects". For a detailed discussion of these effects, see section 2.5 of this report.
- Two packers are installed at an arbitrary distance along the water injection pipe or rod to isolate the test section. Since double packer tests are made sequentially until the length of the hole is tested, the packers must be placed as close as possible to the bottom of the hole at the beginning. They should be lifted a distance equal to the length of the test section (A). Care must be taken with the packer so that "equipment compliance effects" are minimized as much as possible (see section 2.5).
- Water is pumped until steady state is reached (i.e., changes in consecutive readings of pressure and discharge are approximately constant). It is important to get steady state conditions since compliance effects are minimized this way.
- Pressure and discharge readings are recorded at 5 minute intervals.
- The test is repeated until the entire length of the hole is tested.

This test can be run using multiple pressures. The procedure consists of applying pressure in three or more approximately equal steps. For example, suppose that the maximum allowed differential pressure is 50 psi, then the test would be run at 10, 20, 30, 40, and 50 psi. Care must be taken to maintain each pressure step for approximately 20 minutes, and readings should be made at 5-minute intervals. Doing so, the pressure can be raised to the next step until the maximum allowed differential pressure is attained; the process is then reversed, with the pressure being maintained for 5 minutes at approximately the same middle and the lowest pressure steps. In the example shown before, the process for the whole multiple pressure test would be: 10, 20, 30, 40, 50, 40, 30, 20, and 10 psi, approximately. A graph of intake against pressure for all the steps in a multiple pressure test is handy for assessing hydraulic conditions of the tested geological formation.

2.4 Pressure loss in test pipe

Drill rods are the most common intake pipes used in packer tests. NX and NW rods can be used with good results as long as the intake of the test section does not exceed 12 to 15 gallons per minute and the length of the test section is less than 50 feet. It is recommended that a pipe of 1.25 inches of diameter or larger be used in the test. In figures 10-1 through 10-4, plots of head losses per 10-foot section are shown for several water discharges, with different diameter of drill rod and a 1.25 inches diameter rod (the graphs were compiled from experimental data).

Another source of pressure losses comes from the pumping equipment. The most common type of pump for pumping the water is the mud pump. Usually such devices are of the multiple cylinder type with a uniform fluctuation in pressure. It is very difficult, if not impossible, to account for this effect in the calculation of the permeability values. In order to avoid this problem, the use of centrifugal pumps having sufficient capacity to develop back pressure is recommended.

2.5 Interference of packer tests

Pickens et al. [1987] analyzed the effect of borehole pressure history, thermally induced borehole pressure response, borehole and formation skin effects, and equipment compliance effects. Each of these effects will

be presented in some detail.

Borehole pressure history was found to affect significantly by orders of magnitude the value of the hydraulic conductivity. According to Pickens et al [1987], in order to minimize the effect of borehole pressure history, the starting pressure of the test should be substantially different from the formation pressure and the annulus pressure, which is the down hole absolute pressure corresponding to conditions with the borehole fluid level at the wellhead.

Thermally induced borehole pressure response, or the pressure changes resulting from thermal expansion of the borehole fluid, can have an enormous impact on the measurement of the borehole pressure response: a change of temperature in the order of 1 to 2 degrees Celsius, was found to cause a change in pressure in the order of 10 to 100 meters, but these are theoretical results (Pickens et al. [1987]). In real measurements, this effect is moderated by pressure dissipation in the aquifer. This effect is then more critical when measuring a low permeability formation because the pressure dissipation in the aquifer depends on the permeability. Thus, for a high permeability aquifer, the effect of thermally induced borehole pressure response is less critical than in a low permeability aquifer.

Borehole and formation skin effects are perhaps the most important to be considered when performing packer tests. Skin effects are basically the alteration of the permeability of the formation at the well face due, mainly, to the drilling process. In general, skin effects have a great impact (orders of magnitude) on the value of the permeability. This conclusion is common to the work of both Pickens et al. [1987], and Braester and Thunvik [1984]. According to the work of Pickens et al. [1987] (which seems to be more complete on this issue), the value of the permeability can increase due to fracturing and/or erosion, or can decrease due to mineral reactions, mud cake formation, granite flour accumulation, etc. The most common skin effect is drilling mud invasion. The extent of this effect (i.e., the penetration of the mud cake) depends on several factors, like properties of the drilling fluid, porosity of the formation, the ratio of the formation pore size to the size of particulate in the mud, permeability of the formation, permeability of the mud cake, detention time of the mud filling the hole, pressure differential between mud-filled borehole and the formation, etc. However, although the skin effect is known to greatly affect permeability measurements, it is very difficult to quantify the value in real conditions. The determination of the thickness, permeability of a skin, and even its presence in a testing well is nowadays very difficult if not impossible.

3. INTERPRETATION OF PACKER TEST MEASUREMENTS

3.1 Single packer tests

Single packer tests can be analyzed using the Groundwater Manual methodology. This type of packer test is not used often, therefore there are not many methods available in the literature.

3.1.1 Groundwater Manual method

This is the most widely used methodology for the analysis of single packer tests. The estimation of the hydraulic conductivity involves the direct evaluation of an equation involving a simple set of parameters.

3.1.1.1 Assumptions

For the Groundwater Manual method, the following assumptions are made for single packer tests:

- Homogeneous and isotropic formation.
- Confined or unconfined aquifer.
- Measurements and values in English units.

3.1.1.2 Data requirements

For the single packer test, the following data are required to obtain permeability K (remember that English units should be used):

- Radius r of the borehole
- Length of the test section A . For this case, it is the distance between the packer and the bottom of the borehole.
- Depth h_1 from pressure gage to the bottom of the borehole. When using a pressure sensor, use the pressure recorded in the test section prior to pumping as the h_1 value.
- Applied pressure h_2 at the gage, or, in the case of a sensor being used, use the pressure recorded during pumping.
- Steady flow Q into the well. Flow is considered steady when at least three readings at 5-minute intervals are essentially equal.
- Nominal diameter and length of intake pipe between the gage and the upper packer.
- Thickness U of the unsaturated material, or material above the water table.
- Thickness S of the saturated material above a relatively impermeable bed.
- Distance D from the bottom of the test section and the ground surface.

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- Starting test time and time for each of the measurements.
- For streambeds and lakebeds below water, the effective head H is computed as the difference between the elevation of the free water surface in the pipe and the elevation of the gage plus h_2 .
- In the case of a pressure sensor, the effective head should be computed as the difference in pressure before water is pumped into the test section and the pressure readings made during the test.
- The surface area of the test section, a , is the area of the sidewalls of the wet borehole plus the area of the bottom.
- The method is applicable as long as: the value of Q/a is less than or equal to 0.10 (English units) and S/A greater than or equal to 5; also, A/r should be greater than or equal to 10, all measured in English units.

3.1.1.3 Computational Procedure

- Get the head loss coefficient (figure 10-1, 10-2, 10-3 or 10-4 of the Groundwater Manual [USBR, 1985]) = K_1
- Compute $L = K_1 \times (\text{distance from bourdon gage to bottom of pipe})$
- Compute $H = h_1 + h_2 - L$
- Get $T_u = U - D + H$
- If $D > U$ then the permeability can be computed using zone 3 formulae. Otherwise, proceed with the following calculations:
- Compute $X = H / T_u$
- Compute theoretical X' from:

$$w = T_u / A$$

$$X' = 100 - 600 \{1/(wX') [2-100/(wX')] \}^{0.5}$$

Note: this is a recursive equation that requires the use of a numerical technique such as the fixed point algorithm.

- If $X > X'$ then the test is in zone 1 otherwise is in zone 2.
- Once it has been determined in which zone is the test being analyzed, the appropriate formulae should be used:

$$\text{Zone 1: } K = Q / (rHC_u)$$

$$\text{Zone 2: } K = Q / [(C_s + 4)r(T_u + H - A)]$$

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$$\text{Zone 3: } K = Q / [(C_s + 4)rH]$$

$$\text{Where: } C_s = [A/r]2\pi/\ln[A/r]$$

$$C_u = rH/[2\pi(2AH - A^2)][\sinh^{-1}(A/r) - (A/r)]$$

3.2 Double packer tests

The analysis of double packer tests can be made using the Groundwater Manual method. However, since double packer tests are more widely used than single packer tests, alternative methods of analysis are available. In particular, Hvorslev method, Moye method and Dagan method are presented in this report. The method by Dagan [1978] has been mentioned in the literature as the method with a sound theoretical background, and the one that predicts a more accurate value for the hydraulic conductivity (Braester and Thunvik [1984]).

3.2.1 Groundwater Manual method

Double packer test interpretation was developed using the same theoretical assumptions of the single packer test case. The formulae employed for the double packer test data interpretation are very similar to the formulae employed for single packer tests data interpretation.

3.2.1.1 Assumptions

The Groundwater Manual method assumes the following characteristics for the double packer tests:

- Homogeneous and isotropic formation.
- Confined or unconfined aquifers.
- Measurements in English units.

3.2.1.2 Data requirements

For the double packer test, the following data are required in order to compute the hydraulic conductivity K (remember to use English units):

- Radius r of the borehole
- Length of the test section A . In this case, it is the distance between the packers.
- Depth h_1 from pressure gage to the bottom of the borehole. When using a pressure sensor, use the pressure recorded in test section prior to pumping as the h_1 value.
- Applied pressure h_2 at the gage, or, in the case of a sensor being used, use the pressure recorded during pumping.
- Steady flow Q into the well. Flow is considered steady when at least three readings at 5-minute intervals are essentially equal.

DRAFT

- Nominal diameter in inches and length of intake pipe in feet between the gage and the upper packer.
- Thickness U of the unsaturated material above water table.
- Thickness S of the saturated material above a relatively impermeable bed.
- Distance D from the bottom of the test section and the ground surface.
- Starting test time and time for each of the measurements.
- For streambeds and lakebeds below water, the effective head H is computed as the difference between the elevation of the free water surface in the pipe and the elevation of the gage plus h_2 .
- In the case of a pressure sensor, the effective head should be computed as the difference in pressure before water is pumped into the test section and the pressure readings made during the test.
- The surface area of the test section, a, is the area of the borehole wall only.
- In order to apply the method, the value of Q/a must be less than or equal to 0.10 (English units) and S/A must be greater than or equal to 5. Also, the value of A/r must be greater than or equal to 10, all measured in English units. In addition, the length of each packer in this case should be greater than or equal to $10r$.

3.2.1.3 Computational procedure

- Get the head loss coefficient, K_1 , from Figures 10-1, 10-2, 10-3, and 10-4 from the Groundwater Manual [USBR, 1985].
- Compute $L = K_1 \times (\text{Distance from Bourdon Gage to bottom of pipe})$
- Compute $H = h_1 + h_2 - L$
- If $D > U$, then the test is in zone 3 and so the conductivity K can be computed from the formula presented below, without going through the computation of T_w , X and X'.
- Get $T_u = U - D + H$
- Compute $X = H / T_u$
- Compute theoretical X' from:

$$w = 1/(T_u/A)$$

$$X' = 100 - 600 \{1/(wX') [2-100/(wX')] \}^{1/2}$$

Note: it is a recursive equation that requires the use of a numerical technique such as the fixed point algorithm.

DRAFT

- If $X > X'$ then the test is in zone 1 otherwise is in zone 2.
- Once it has been determined in which zone is the test being analyzed, the appropriate formulae should be used:

$$\text{Zone 1: } K = 2Q / [(rC_s) (T_u + H - A)]$$

$$\text{Zone 2 and Zone 3: } K = Q / [C_s r H]$$

$$\text{Where: } C_s = [A/r]2\pi/\ln[A/r]$$

3.2.2 Dagan method

Braester and Thunvik [1984] showed that the method presented by Dagan [1978] estimated hydraulic conductivity more accurately than the methods proposed by Hvorslev [1951] and Moye [1967]. The method is not as straightforward as the others presented in the Ground Water Manual, and requires the solution of a system of linear equations with a dense matrix of dimension 20 (Dagan [1978]). However, it can be solved easily using a spreadsheet.

3.2.2.1 Assumptions

The following assumptions are incorporated into the Dagan [1978] method:

- Unconfined aquifers.
- Isotropic formations.
- Water is injected into a double packer system set-up.
- Measurements are taken at steady state conditions.

3.2.2.2 Data requirements

The method proposed by Dagan [1978] needs the following input data:

- The well radius, r .
- The length of the test section, A .

Note: This method is accurate if and only if $A > 50r$. If this condition is not satisfied, the results may not be accurate (Dagan, 1978).

- The water discharge, Q .
- The effective head, H .
- Length of the test section (distance between packers), A .

DRAFT

- Water table depth, U.
- Distance from ground surface to lower packer, D.
- Note that in this method the units are not restricted, so either English or SI units can be used in a consistent way.

3.2.2.3 Computational procedure

The model proposed by Dagan (Dagan [1978] and Braester and Thunvik [1984]) consists of distributing the water discharge Q induced into the well into a finite number of discharges q_i so that:

$$[M]^T \{u\} = \{\Delta P\}$$

$$Q = \Delta L \sum_{i=1, \dots, N} q_i$$

$$u_i = q_i / K$$

$$\Delta P = H + I$$

$$\begin{aligned} \text{Where: } I &= 0 \text{ if } U < D \\ I &= U - D + A/2 \text{ otherwise.} \end{aligned}$$

So the method basically requires the assembling of a system of simultaneous equations to get the values for u_i , solve the system, and then compute K from:

$$K = Q / [\Delta L \sum u_i]$$

$$\Delta L' = A/N$$

The definition of the elements of the matrix M is as follows:

$$M_{ij} = 1/4\pi \ln \left\{ \frac{[(|a_{ij}| + a_{ij} + r^2/2|a_{ij}|) / (|a_{ij}-\Delta L| + a_{ij} - \Delta L + r^2/2|a_{ij}-\Delta L|)]}{[(|c_{ij}-\Delta L| + c_{ij} - \Delta L + r^2/2|c_{ij}-\Delta L|) / (|c_{ij}| + c_{ij} + r^2/2|c_{ij}|)]} \right\}$$

where:

$$a_{ij} = (j - i + 0.5)\Delta L$$

$$c_{ij} = (j + i - 0.5)\Delta L - 2D$$

$$N = 20 \text{ \underline{always}} \text{ (this is to avoid numerical instability problems, according to the recommendations of Dagan [1978])}$$

3.2.3 Other methods: Hvorslev and Moye equations

Hvorslev [1951] and Moye [1967] methods are presented in good detail by Braester and Thunvik [1984]. The two methods are very straightforward, and consist mainly of a simple equation, including a natural logarithm.

DRAFT

According to Braester and Thunvik [1984], Hvorslev and Moya methods tend to overestimate hydraulic conductivity.

3.2.3.1 Assumptions

The methods proposed by Hvorslev [1951] and Moya [1967] are studied in the work by Braester and Thunvik [1984]. They consist of a simple equation that computes the hydraulic permeability, K , based on the discharge, Q , the induced pressure, ΔP , and the geometric characteristics of the aquifer. Both methods assume that:

- The tested region is located at such depth that the water table is not affected by the injection of small volumes of water for a relatively short period of time.
- According to the previous assumption, the phreatic level is assumed to be a fixed boundary.
- The aquifer is unconfined, homogeneous and isotropic.

3.2.3.2 Data requirements

The data required for Hvorslev and Moya methods are:

- The radius of the borehole, r .
- Water discharge, Q .
- Effective head, H
- Length of the test section, A .
- Any consistent units are valid.

3.2.3.3 Computational procedure

The computational procedure for Hvorslev and Moya methods is very simple; basically, given the information mentioned in the previous section, the hydraulic conductivity K is computed with the following equations:

- Hvorslev method:

$$K = Q \ln [A/2r + \{ 1 + (A/2r) \}] / (2\pi AH)$$

- Moya method:

$$K = Q \{ 1 + \ln(A/2r) \} / (2\pi AH)$$

3.3 Application examples

Two application examples are presented, one relating a single packer test, and the other for a double packer test. The example of the single packer test is the same as one of the examples in the Ground Water Manual [USBR, 1985]. The double packer test example is one of the cases from the Harza Engineering report [1984]

3.3.1 Single packer

The example that will be presented in the following subsections was taken from the Ground Water Manual [USBR, 1985].

3.3.1.1 Data Provided

For this example, assume the following data was given:

$U = 75$ ft, $D = 25$ ft, $A = 10$ ft, $h_1 = 32$ ft, $h_2 = 57.8$ ft,
 $Q = 0.045$ ft³/s (cfs), $L = 1.7$ ft, $r = 0.5$ ft.

3.3.1.2 Solutions

- Applying the Groundwater Manual Method, we get:

$$K = 1.5 \times 10^{-5} \text{ ft/s}$$

Now, as a way to compare the result obtained for the only methodology available specific for single packer tests, we obtained the conductivity using the methods for double packer tests:

- Moye method gives: $K = 2.7 \times 10^{-5}$ ft/s

- Hvorslev method gives: $K = 2.5 \times 10^{-5}$ ft/s

3.3.1.3 Comments on the Results

Looking at the results we can see that the highest estimate for the conductivity is the Ground Water Manual value. Hvorslev and Moye formulae give a similar result, which is lower than the Ground Water Manual value. Single packer tests are not very common in practice; double packer tests, on the other hand, are of much frequent occurrence.

3.3.2 Double packer

The example that will be presented in the following subsections was taken from the Harza Engineering report [1987]

3.3.2.1 Data Provided

For this example, assume the following data was given:

$U = 95.2$ ft, $D = 490.0$ ft, $A = 21.0$ ft, $h_1 = 35.0$ ft, $h_2 = 42.0$ ft,
 $Q = 0.000446$ ft³/s (cfs), $L = 0.0$ ft, $r = 0.125$ ft.

3.3.2.2 Solutions

- Applying the Groundwater Manual Method, we get:

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$$K = 2.96 \times 10^{-6} \text{ ft/s}$$

- Moya method gives: $K = 3.14 \times 10^{-6} \text{ ft/s}$
- Hvorslev method gives: $K = 2.97 \times 10^{-6} \text{ ft/s}$

The application of Dagan method is quite complicated and was thus performed using a spreadsheet. The spreadsheet was configured so that with two simple macros it is possible to compute the solution fast and efficiently, changing just a few values in the cells of the spreadsheet file. The result obtained in this case was:

- Dagan method gives: $K = 2.76 \times 10^{-6} \text{ ft/s}$

3.3.1.3 Comments on the results

Looking at the results we can see that the lowest estimate for the conductivity is the Ground Water Manual value. Hvorslev and Moya formulae give a similar result, which is higher than the Ground Water Manual value. The value for Dagan method is the lowest of all the others, but very close to the values provided by Hvorslev and the Ground Water Manual. This result is not surprising, however. The fact that Hvorslev, Moya and the Ground Water Manual methods tend to overestimate the value for the hydraulic conductivity has been noted by Braester and Thunvik (1984). The requirement of $A/r > 50$ is satisfied. Therefore, the results are accurate enough using Dagan method.

APPENDIX III

PAPER TO BE PRESENTED AT THE
8TH INTERNATIONAL SYMPOSIUM
ON FREIGHT PIPELINES

An Investigation into the Breakage Characteristics of Fly Ash Plugs during Pneumatic Backfilling of Abandoned Coal Mines

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Abstract

The minimum pressure required to break a stationary plug of fly ash with a pulse of air was measured in an experimental apparatus. Plugs of various lengths were prepared in seven inch clear acrylic tubing. The plugs were subjected to bursts of pressurized air and the results were observed. Particles in the upper portion of the plug were displaced in a repeatable pattern. The backside of the plugs were severely fractured. A linear relationship was found to exist between plug length and break pressure. A pressure of approximately 0.5 psi/ft was required to break the plugs. At pressures ten to twenty times higher than the minimum pressure, the entire upper portion of the plug was sheared off. The remaining bottom portion of the plug formed into a wavy pattern.

Introduction

Pneumatic stowing is a well established practice for preventing mine subsidence. Subsidence occurs when the structure of the mine begins to deteriorate and the mine collapses. Filling the mine with support material can prevent this process from occurring. Fly ash serves as an excellent support material, because it form forms a cement when mixed with water. Furthermore, by using the fly ash as backfilling material the cost of disposal is averted.

In a typical operation, the backfilling process begins from the end of a mine shaft [1]. The transport line is then withdrawn as the mine is filled in. Because entrance to an abandoned mine is too dangerous, a new technique for backfilling sealed mines is being developed.

Bore holes will be drilled to allow the transport of fly ash, and ventilation of air. This technique offers challenges not found with conventional stowing methods. To reduce drilling costs it is necessary to determine the maximum conveying distance. The material must be conveyed in broad channels where the cross-sectional area will limit the conveying velocity which can be achieved. At these low velocities, saltation and plug formation are a serious concern.

The breaking of stationary plugs has not been explored in great detail. Tsuji, Morikawa and Honda [2] found that for coarse particles, the critical pressure required to break a plug is proportional to plug length. These investigators determined the critical pressure by gradually increasing the air flow rate through a plug and measuring the pressure drop when the plug was broken. Furthermore, the critical pressure decreased with an increase of pipe diameter.

The breaking pressure will undoubtedly be dependent upon the properties of the plug material. Cohesiveness, particle size distribution, shape and air retention are some of the properties which determine the breaking pressure. [3] In addition to the properties mentioned above, the structural integrity of each plug must be considered.

Current work concerns the breaking of fly ash plugs in seven inch clear acrylic tubing. The pressure required to break the plugs is measured and the flow patterns are observed. This investigation will provide a basis for determining the feasibility a new pneumatic stowing technique.

Experimental

Figure 1 shows the configuration of the apparatus used to test plug breakage. Compressed air is fed into the air reservoir. The reservoir is a five foot section of four inch carbon steel pipe. The pressure is measured by a gauge near the air inlet. The reservoir is connected to the test section by a fast acting, two inch, pneumatically actuated valve. Opening this valve allows the pressurized air to enter the test section.

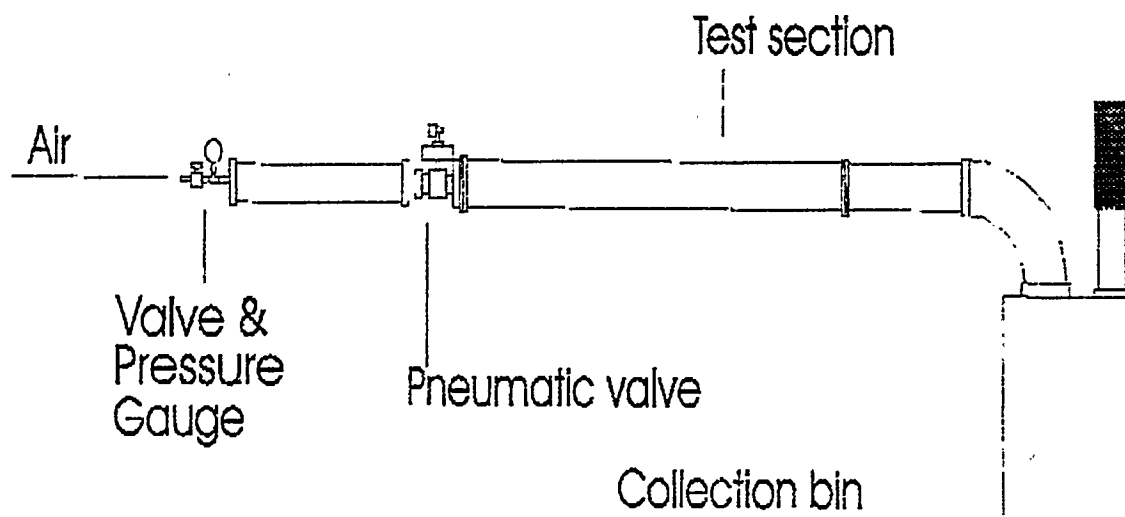


Figure 1. Experimental setup

The test section consists of a twelve foot long section of seven inch clear acrylic tubing. The tubing is flanged at both ends. The flanges permit connection of the test section to the pneumatic valve and the collection bin.

To form the plugs, the test section is removed and rotated vertically. One of the ends is sealed with a cap and plug retainer. The retainer prevents the plug from filling the entire end of the tube. It also serves to produce a uniform void volume between the plug and the valve. Fly ash is poured into the tube while the test section is seated on a scale. When the desired weight of ash has been added to the tube, the plug is allowed to settle. To enhance the settling rate, the tube is tapped repeatedly with a dead-blow hammer. The

settling process continues until the bulk density reaches an approximate value of 64.4 lb/ft³. This bulk density is approximately equal to what would be achieved if the material would be allowed to settle naturally for one hour.

When the fly ash has settled, the test section is attached to the apparatus and the air reservoir is pressurized. The ends of the plug deform leaving the top section of the plug at its minimum width. This width is then recorded as the plug length. When the pressure measurements have been recorded, the pressure is released into the test section. Before the air reaches the plug it expands through a short length of connecting pipe and a void volume before the plug. The effect of the pressure wave on the plug is observed and recorded. The pressure exerted on the face of the plug is calculated from the expansion volume.

Results

The minimum break pressure of fly ash plugs was experimentally determined for plugs of 2, 4, and 8 feet in length. The results of these experiments are given in figure 2. A linear relationship appears to exist between break pressure and plug length. The critical break pressure is 0.5 psi/ft of plug.

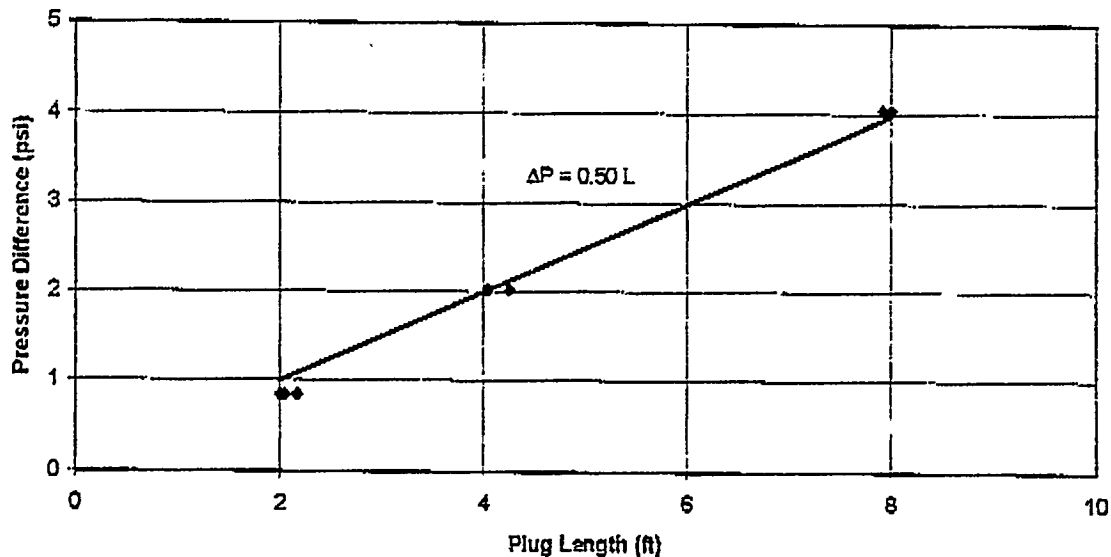
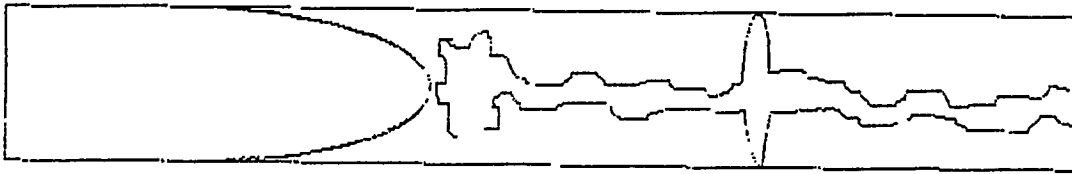


Figure 2. Pressure Difference vs Plug Length for Stationary FBC Fly Ash Plugs (Inside Diameter = 6.5 in)

Figure 3 shows the typical fracture pattern at the minimum break pressure. The top view indicates a parabolic indentation on the front side of the plug. This shape was seen in both broken and unbroken plugs. Beyond the parabola, a roughly circular hole forms which is typically about two inches in diameter. This hole is the starting point of a path which meanders through the remainder of the plug. Fissures frequently form along the meandering path. These fissures may extend deep into the plug. A fissure is evident in the side view of the broken plug. On plugs which did not break, the air seemed to be held in the parabolic area.

Top View:



Side View:

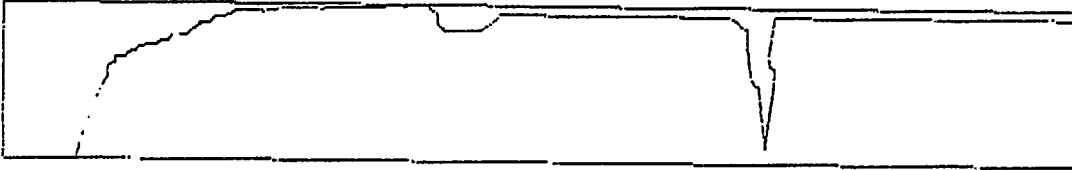


Figure 3. Top and side views of a plug broken at its critical pressure.

Several tests were also performed at much higher pressures. Table 1 indicates the pressures and plug lengths corresponding to these experiments. Figure 4 shows the break

<i>Case</i>	<i>Plug Length (ft)</i>	<i>Pressure Difference (psi)</i>
1	1.33	38.9
2	3.17	23.3
3	3.25	38.9
4	3.58	31.1
5	5	38.9

Table 1. High pressure plug breakages

Side View:

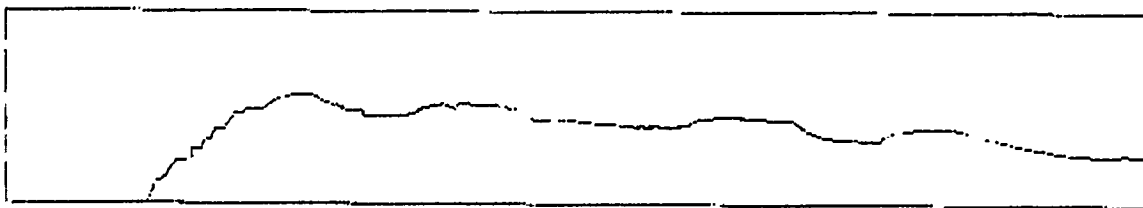


Figure 4. Side view of a plug broken at a high pressure.

pattern to be significantly different at higher pressures. In case one, the plug was short and fragile. When it was hit with the blast of air, the plug was completely demolished. In cases 2-5, only the bottom half of the plug, in a wavy formation, remained after pressure was applied. The plugs were moved about six inches before breakage in cases 3 and 4.

The fly ash has excellent air retention properties. To insure that the plugs were similarly compact, the bulk density of each plug was calculated. The mean bulk density for each plug length was calculated and is reported in Table 2.

<i>Approximate Plug Length (ft)</i>	<i>Mean Bulk Density (lb/ft³)</i>
2	64.1
4	64.4
8	64.6

Table 2. Mean bulk densities of plugs of various lengths.

Discussion

The greatest challenge in this study was the creation of uniform plugs. The fly ash is highly aeratable. Its ability to retain air made it difficult to form plugs which did not develop air pockets as they settled. Striking the tube with a dead-blow hammer helped to remove the air pockets somewhat, but when that failed, the plugs were unusable.

The plug forming problem was further complicated by the necessity of rotating the tube from a vertical to a horizontal position. During rotation, the ends of the plug deforms somewhat. Because of this, it was extremely difficult to predict the plug length before rotation. Sometimes, the rotation of the tube caused the formation of cracks in the plug. When cracks were found, the plug was considered unusable.

This method of building plugs suffers from another problem. Since the plug is allowed to settle in a vertical orientation, the compaction is not uniform across the length of the plug. This situation does not ideally represent a plug which might form during the backfilling process. In an effort to better represent real plug formation, attempts were made to extrude plugs from a blow tank into horizontal pipes. Two difficulties prevented the use of this method. First, it was difficult to release the air from the fly ash. Thus, the plugs would be only minimally compacted and settling would produce gaps between the pipe and plug. Secondly, the only pressure vessel available had two inch diameter outlet. The step from two inches to six inches disrupted the homogeneity of the extrusion. The entire cross-section of six inch pipe could not be filled after the expansion. The extrusion method was abandoned because of these difficulties.

To show that the pressure applied to the plug was the minimum, pressures less than that were attempted and shown to hold for a longer period of time. At pressures slightly less than the critical pressure, the plug would initially hold, but after approximately one minute, the continued pressure would finally seep through the plug and break in apart. It was not considered to be a break due to a pressure blast unless the plug broke in less than five seconds.

Previous work in plug breaking does not lend itself to direct comparison to this investigation. In the earlier investigations, an equilibrium developed between the air permeating through the plug and the pressure drop. [1] When the air flow rate reached a critical value, the plug would deteriorate. In this study, the plug is subjected to a pulse of pressurized air. A shock wave distorts the front side of the plug forming the parabolic pattern depicted in Figure 4. If the pressure is above the critical pressure, the air continues to break through in a meandering pattern following the path of least resistance. Because of these differences, the data from the present study can not be reliably compared to previous investigations.

Conclusions

The experiments have shown that fly ash plugs can be broken with pulses of pressurized air. The minimum pressure required to break the plugs is proportional to the plug length. At the minimum pressure only particles in the upper portion of the plug were

noticeably displaced. The air surged into the plug leaving a parabolic pattern in its wake. The air penetrated the remainder of the plug while creating fissures along its meandering path. At pressures significantly higher than the minimum break pressure, a different flow pattern is observed. With pressures ten times the minimum pressure, the top portion of the plug is sheared off. The remaining material is distributed across the bottom of the pipe in a wave like pattern.

References

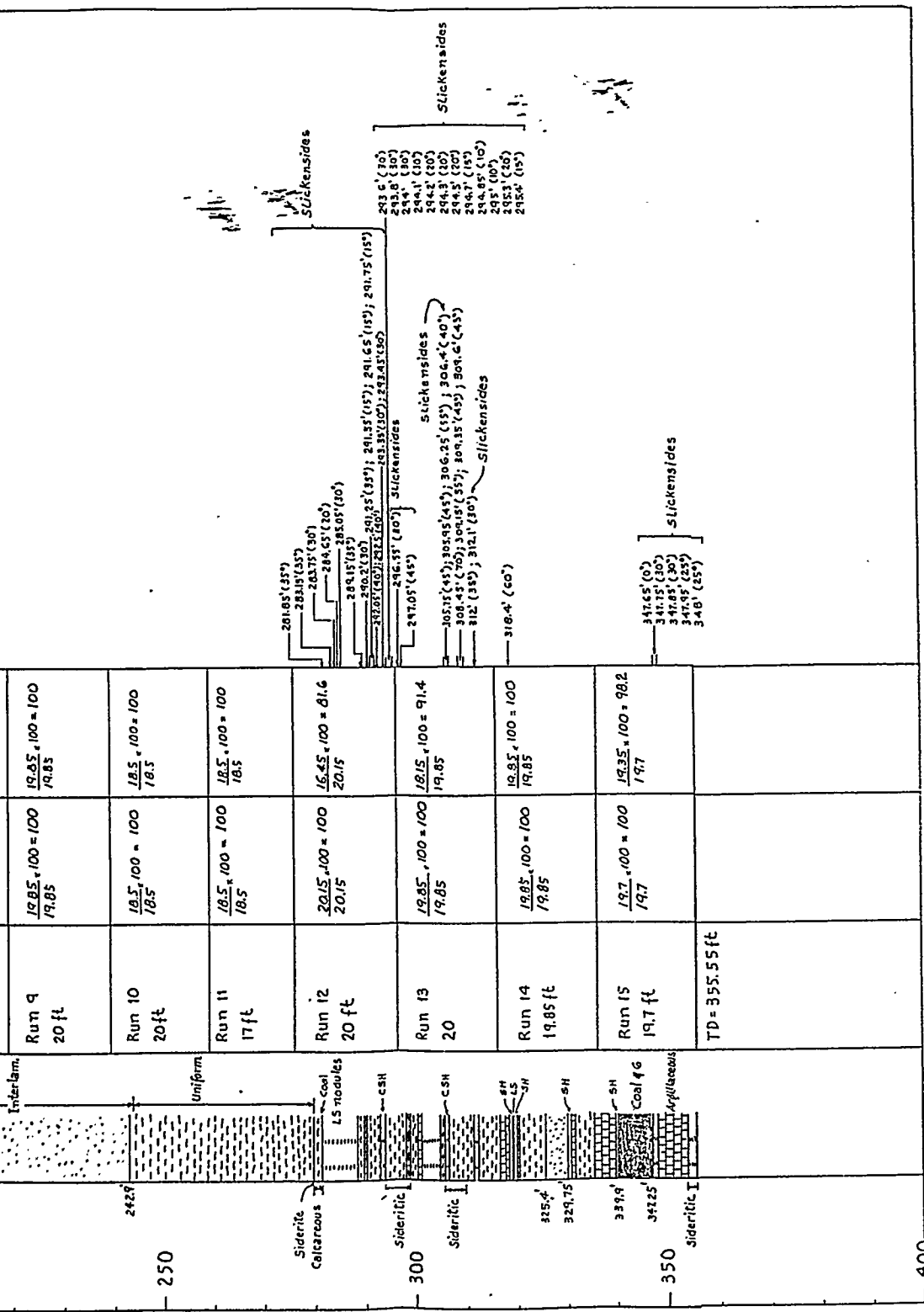
1. Peter, G. Measurements Made On Pneumatic Stowing Machines for Determining the Relationship Between Stowing Efficiency, Length of Pipe Quantity of Air and Pressure. *Transl. 3780, National Coal Board Transl. A. 261, from Glukhauf year 88, v. 33/34, (1952), 807-819.*
2. Tsuji, Y., Morikawa, Y., and Honda, H. A Study On Blowing Off a Stationary Plug of Coarse Particles in a Horizontal Pipe. *Journal of Powder & Bulk Solids Technology.* 3 (1979) 4, 30-35.
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APPENDIX IV

GEOTECHNICAL CORE LOG

Peabody 10 - Section 10 - Christian Co. - Core size: 3" Preliminary Geotechnical Core Log

Depth (ft.)	Lithology	Drilling Data	Core Recovery (%)	RQD (%)	Fractures
0					
50	61.5' weathered Dark SH Coal LS nodules Loss (4.3) Loss (0.2) SH CSH CSH LS (0.3) LS (0.3) Calcareous Siderite LS nodules Calcareous Calcareous II Interlam. Loss (0.4)	Run 1 17 ft Run 2 20 ft Run 3 19.5 ft Run 4 21 ft Run 5 20 ft Run 6 20 ft Run 7 20 ft Run 8 20 ft	$\frac{12.5}{17} \times 100 = 73.5$ $\frac{20}{20} \times 100 = 100$ $\frac{18.2}{19.5} \times 100 = 96.9$ $\frac{20}{20.9} \times 100 = 95.7$ $\frac{20.1}{20.1} \times 100 = 100$ $\frac{19.6}{20} \times 100 = 98$ $\frac{20}{20} \times 100 = 100$ $\frac{20.5}{20.5} \times 100 = 100$	$\frac{12.5}{17} \times 100 = 73.5$ $\frac{20}{20} \times 100 = 100$ $\frac{18.4}{19.5} \times 100 = 94.4$ $\frac{20}{20.9} \times 100 = 95.7$ $\frac{20.1}{20.1} \times 100 = 100$ $\frac{19.6}{20} \times 100 = 98$ $\frac{20}{20} \times 100 = 100$ $\frac{20.5}{20.5} \times 100 = 100$	96.9' (40°) 97.9' (45°) 101.7' (10°) 102.05' (23°) 102.2' (25°) Slickensides 191.3' vertical
100					
150					
200					



NK
7.7.1995
JSGS

LEGEND:

- Carbonaceous shale (black)
- Clay shale
- Coal
- Limestone
- Sandstone
- Shale
- Siltstone
- Soil & Glacial till
- CSH - Clay shale
- Interlam. - Interlaminated
- LS - Limestone
- SH - Shale